BENEFICIATION OF RARE EARTH MINERALS

FROM BOKAN MOUNTAIN – DOTSON RIDGE ORE

By

Purevbaatar Narantsetseg

RECOMMENDED:

Dr. Guven Akdogan

Dr. Tathagata Ghosh
Advisory Committee Co-Chair

Dr. Rajive Ganguli
Advisory Committee Co-Chair

Dr. Rajive Ganguli, Chair
Department of Mining and Geological Engineering

APPROVED:

Dr. Doug Goering
Dean, College of Engineering and Mining

Dr. John Eichelberger
Dean of the Graduate School

12/23/14
Date

BENEFICIATION OF RARE EARTH MINERALS FROM BOKAN MOUNTAIN – DOTSON RIDGE ORE

A

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Purevbaatar Narantsetseg

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Abstract

The purpose of this research work was to study the beneficiation of rare earth ore of the Bokan Mountain – Dotson Ridge deposit, located near Ketchikan, Alaska. Rare earth element (REE) composite ore samples from the Bokan Mountain – Dotson Ridge deposit were tested using gravity concentration, magnetic separation, flotation, and leaching techniques to separate the REE. The composite ore sample was a product of a preliminary x-ray sorting process. Qualitative electron microprobe analysis of the ore showed that most of the REE minerals in the ore were silicate minerals. Since the electron microprobe analysis samples were coated with carbon during sample preparation, the carbon element was inactivated for analysis. Because of this, carbonate compounds of minerals' particles could not be detected. 95% of the REE mineral particles appear to be smaller than about 10 μm in size (about 100 μm^2 in area).

For the gravity concentration, light rare earth elements (LREE) and heavy rare earth elements' (HREE) individual elemental recovery values were in the ranges of 49.6-52.8% and 46.3-48.8%, respectively, at 25% of mass yield. In order to separate a larger amount of the REE, a wet high intensity magnetic separation (WHIMS) test was carried out on tailings of the gravity concentration tests. The HREE individual elemental recovery values ranged from 56.3-63.1% at 37% mass yield, while LREE individual elemental recovery values were in the 57.9% - 59.1% range. For the combined gravity and magnetic separation processes, the net individual elemental recovery values of the LREE and the HREE were in the range of 79.6-80.5% and 76.5-80.9%, respectively. The combined mass yield of the gravity and magnetic separation processes was 53%.

Direct leaching tests conducted on the composite ground ore feed yielded high individual elemental recovery values of 90-92% of the LREE. The HREE individual elemental recovery values ranged from 56.5-87.3%. In the leaching, 20% HCl was used in the 1st and 2nd stages with a duration of 2 hrs in each stage at 90°C. The solid percentage of the leach slurry was 20% w/w.

The composite ground ore sample was tested in conventional flotation using a 2.0 L capacity Denver cell. In the flotation, 0.05 kg/tonne of Cytec Aero 6493 collector, 0.05 kg/tonne of Cytec Aero Froth 88, and 0.1 kg/tonne of sodium metasilicate as a depressant were used. Pulp pH was set around 9. Results showed individual elemental recovery values in the range of 44.6-50.4% for the LREE. The HREE individual elemental recovery values ranged from 27.9-44.5%. The mass yield of the flotation was 23%. The flotation recoveries reported here are significantly lower than what was achieved previously. For the Leach after flotation process, leaching was conducted on the first concentrate of flotation. Individual elemental recovery values of the LREE and HREE were 94.7-96.5% and 61.1-90.5%, respectively. The concentrate was leached using 20% HCl in both the 1st and 2nd stages, with a duration of 2 hrs in each stage at 90°C. Flotation/Leach process net recoveries of LREE by individual elemental values ranged from 42.2-48.5%. HREE net recoveries by individual elemental values ranged from 17.1-41.4%.

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Chapter 1 Introduction and Objectives

Beneficiation of REE is different, and more challenging, than for other metals because for most ores, there is only one (or very few) main element to be recovered, whereas for REE, there are several. This thesis highlights the challenges as it deals with 15-17 value elements, their co-existence in mineral particles, their individual elemental recoveries, and four different mineral processing strategies.

REE include the fifteen lanthanide elements, coupled with yttrium (Y). The lanthanide elements are divided into two groups: the light rare earth elements (LREE), which include lanthanum (La) to europium (Eu) (atomic numbers: 57-63), and the heavy rare earth elements (HREE), which include gadolinium (Gd) to lutetium (Lu) (atomic numbers: 64-71). Yttrium (atomic number: 39) is usually included in HREEs due to their chemical and physical similarities.

As noted in (Jordens, et al., 2013) demand for REE has increased in recent years due to their wide-ranging applications from magnets to communication systems. Due to a variety of reasons, China, the primary supplier of REEs, reduced their REE exports recently, resulting in other countries looking to develop their own REE resources. Examples of this include the Bokan Mountain mine in Alaska, the Mountain Pass mine in California, and Mount Weld in Australia.

The Bokan Mountain – Dotson Ridge property is a potential REE resource in Alaska. It is located on southern Prince of Wales Island, approximately 60 km (38 miles) southwest of the city of Ketchikan. Ucore Rare Metals Inc. owns the property. The mineralogical properties of the orebody are discussed in the next chapter.

The purpose of this research work is to study beneficiation of the REE from the ore of the Bokan Mountain – Dotson Ridge deposit using mineral processing techniques such as gravity concentration, magnetic separation, flotation, and leaching. The composite ore sample was a product of a preliminary x-ray sorting process. Qualitative electron microprobe analysis of the ore was also conducted to explore the composition of the rare earth minerals (REM). It showed that most of the REE minerals in the ore were silicate minerals. 95% of the REE mineral particles in

the ore might be smaller than about 10 μm. Carbonate compounds of mineral particles were not detected, as they were suppressed from detection because the samples were coated with carbon in preparation for electron microprobe analysis. Gravity concentration was conducted on the ore. A wet high intensity magnetic separation (WHIMS) test was carried out on tailings of the gravity concentration tests. Direct leaching tests were conducted on the ore. The ore sample was also tested by conventional flotation. After the flotation, leaching was conducted on the first concentrate (concentrate 1) of flotation.

Chapter 2 Literature Review

2.1 Gravity separation of REE ore

Gravity concentration methods can separate materials of different specific gravities by their relative movement in response to gravity and other forces, including the resistance to motion exerted by a fluid such as water or air. It is important for separation and concentration by gravity that there is a distinct specific gravity difference between the various components, given as the ratio:

$$SG\ Ratio = \frac{\rho_{sH} - \rho_w}{\rho_{sL} - \rho_w} \tag{2.1}$$

where ρ_{sH} – specific gravity of heavy component, ρ_{sL} – specific gravity of light component and ρ_w – specific gravity of water.

Generally, for a ratio of 2.5 or higher, gravity separation is relatively easy and the separation of particles down to 200 mesh is possible. When the ratio is 1.75, separation is effective down to 65 mesh only, while for 1.50 it is effective only down to 10 mesh. Below 1.25, only material above 12.5 mm in size can be separated by gravity, and separation cannot be in water. At this larger particle size, another fluid heavier than water is needed for separation (Gill, 1991).

The ratio of the specific gravity difference among bastnaesite, fluorite, and quartz was analyzed using pure minerals (Kim, et al., 2010). The specific gravities of fluorite, bastnaesite and quartz are 3.18, 5.21, and 2.65 respectively. Table 2.1 indicates the ratio of the specific gravity difference between the pure minerals in distilled water.

Table 2.1 The ratio of the specific gravity difference among the pure minerals (Kim, et al., 2010)

Pure minerals	Ratio
Bastnaesite and fluorite	1.93
Bastnaesite and quartz	2.55

The ratio between bastnaesite and fluorite is 1.93, and that between bastnaesite and quartz is 2.55. Therefore, gravity separation of fluorite and quartz from bastnaesite has good potential, with separation of quartz from bastnaesite being easy (Kim, et al., 2010).

Gravity separation test on rare earth oxides (REO's) from the Mt Weld deposit in Western Australia was conducted by Guy, et al., 2000. The CZ-type (limonitic siltstone) ore has the highest REO grade and is the most extensive ore type in the Mt Weld lanthanide sequence. About 46 percent of the REO in crushed CZ ore is coarser than 212 μm. Given this distribution and the difference in specific gravity between the rare earth minerals present, such as monazite (4.9 - 5.2), cheralite (5.4) and cerianite (7.2), and the main gangue mineral goethite (4.3), gravity pre-concentration appeared to be a promising step towards increasing overall REO recovery. Previous work had indicated that about 50% of the coarse REOs could be concentrated effectively by gravity methods, but concentrate grades could not be raised above 32% REO; this was thought to be connected to difficulties in dispersing dried Mt Weld ore. Hindered settling and superpanner tests were carried out on discrete size fractions of CZ ore prepared using the modified grinding procedure. The ore had 23.2% REO, 44.1% Fe_2O_3, and 1.62% CaO. The REO consisted of CeO_2, La_2O_3, Nd_2O_3, and Pr_6O_{11}. The results showed that in a single pass 24.3% of the REO in fresh feed can be concentrated in 10.4% of the weight into a gravity concentrate assaying 51.9% REO. The size-by-size results showed that high concentrate grades (+50% REO) were consistently obtained across the size range 250 -1680 μm, falling off only below 250 μm. (Guy, et al., 2000)

2.2 Magnetic separation of REE ore

The basis for magnetic separation is a magnetic field that is utilized to create differential movement of mineral particles. The effectiveness of the separation depends on the difference in magnetic susceptibility of minerals (Gill, 1991). Magnetisation curves of bastnaesite and monazite were linear, meaning the minerals are paramagnetic. On the other hand, fluorite and quartz are diamagnetic. Therefore, it is possible to separate paramagnetic rare earth minerals such as bastnaesite and monazite from diamagnetic fluorite and quartz by magnetic separation (Kim, et al., 2010).

2.3 Froth flotation of REE ore

Different minerals have different abilities to adhere to air bubbles in a mineral-water slurry. This property is exploited in froth flotation to physically separate particles. The air bubbles carry the attached particles to the surface of the slurry, separating them from the particles that remain in the slurry. Chemicals can be used to strengthen or weaken the ability of a mineral surface to attach itself to an air bubble, thereby extending the applicability of this technique to separate a wider range of minerals. (Kawatra, 2011)

A new collector for bastnaesite flotation, modified hydroxamic acid (MOHA), had been designed in the laboratory (Ren, et al., 1997). The flotation recovery of pure bastnaesite mineral with MOHA vs. the pH value and the concentration of MOHA is shown in Figure 2.1 a and b, respectively. As seen in Figure 2.1, with increase in pH, flotation recovery suddenly increases until pH=8.5. At pH 8.5 - 9.5, the flotation recovery achieves its maximum and then goes down in strong alkaline media. The flotation recovery rises with increase of concentration of MOHA. Flotation recovery was 90% at 1.0×10^{-3}mol/l of MOHA, and 95% at 5.0×10^{-3}mol/l of MOHA. (Ren, et al., 1997)

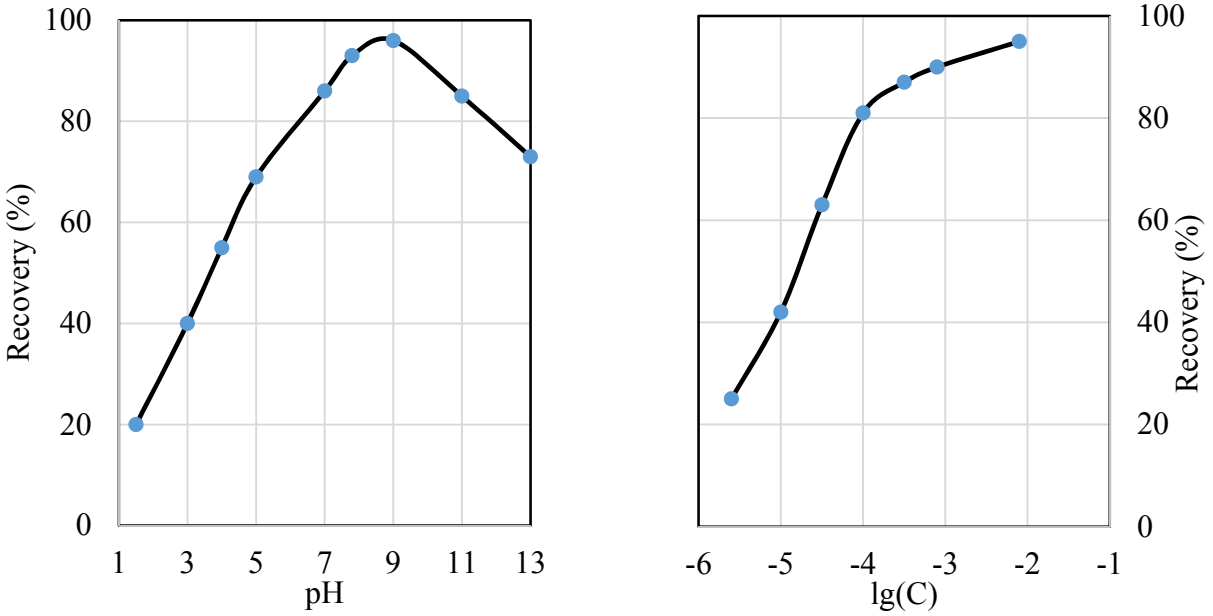

Figure 2.1 The effects of pH value and MOHA concentration C (mol/liter) on the flotation recovery of bastnaesite (Ren, et al., 1997)

The weakest complexes of hydroxamates are combined with alkaline-earth metal cations (Ca^{2+}, Ba^{2+}, Sr^{2+}) and rather strong complexes are combined with the highly charged multivalent rare-earth cations and with the transition elements such as Nb, Ti, V, Mn, Zr, Hf, Ta. The strongest complexes are formed with Fe^{3+}, Ta^{5+} and Nb^{5+}. It has also been informed that differences in the stability constants of complexes formed with lattice cations of the minerals to be separated in a carbonatite ore (such as lanthanides from alkaline earth-containing gangue minerals) are much greater for hydroxamic acid-based collectors than for carboxylic acids (fatty acids). It is anticipated, therefore, that alkyl hydroxamates should be more selective in the separation of bastnaesite ore deposits of the type that occur at Mountain Pass, California (Fuerstenau D.W. and Pradip, 1991).

According to (Gupta C. K. and N. Krishnamurthy, 2004), the ore at Mountain Pass is ground to 100% passing 150 mesh for flotation. The ore contains bastnasite, and averages about 7% rare earth oxide (REO) content. The flotation feed is conditioned by steam, soda ash, sodium

fluosilicate, sodium lignosulfonate, and steam-distilled tall oil. Flotation is conducted hot and produces a bastnasite concentrate that contains 60% REO.

Fatty acids have conventionally been the collector of choice in bastnaesite flotation because of their well-known availability and use in mineral flotation in general. However, they have been known to be unselective for bastnaesite flotation, requiring elevated temperatures, and the addition of large volumes of depressant to achieve a reasonable separation (Bulatovic, 2007), (Bulatovic, 2010), (Jianzhong, et al., 2007).

The tailings of flotation of quartz monzonite ore were processed using gravity separation, magnetic separation and flotation (Aplan, 1988). The monazite was beneficiated from wolframite and other 'black minerals' by flotation in order to produce tungsten as the tailings of flotation. Wolframite and other 'black minerals' were in both a monazite beneficiation stream and in a tungsten stream where small quantities of monazite were floated from the wolframite. The flotation process for monazite was conducted using a patent (Cuthbertson, 1952). The monazite concentrate from this process had 12.5% P, so this concentrate was 94% pure. The collector which was used in flotation was coco amine acetate (Armac C). The depressant of wolframite/black mineral was causticized starch. Sulfuric acid was added to pH 1.4 and the slurry was heated to 95°F (Aplan, 1988).

2.4 Leaching of REE

Metals are often found in nature within metal-bearing minerals. The metal can be extracted from the mineral by breaking chemical bonds. Thus, chemical reactions are needed to separate metals from minerals. In order to break a chemical bond, a more favorable alternative must be provided. One key to leaching is contact between the metal and the water. After contact, a reaction is needed to solubilize the metal. The reaction converts the metal to an ion form to make it soluble. After the metal is solubilized, it can be processed in solution. (Free, 2013)

In the Mountain Pass mine, to further enhance grade of the bastnaesite, the flotation concentrate is leached with 10% hydrochloric acid (HCl) at pH 1 in 1.8 m diameter× 2.7 m leach tanks. This

process dissolves the residual carbonates and increases the grade to 68-72% REO. (Fuerstenau D.W. and Pradip, 1991)

2.5 Mineral processing of Bokan Mountain – Dotson Ridge REE ore

2.5.1 Mineralogical property

The Bokan Mountain – Dotson Ridge deposit has complex REE mineralogical characteristics. There are 4.7% REE silicates, 0.9% REE carbonates, 0.34% REE phosphates and 0.6% REE oxides in ore of the deposit. Two major REE silicates are zircon ($ZrSiO_4$) and allanite ($Ca_2Al_3(SiO_4)_3(OH)$). A fraction of zirconium (Zr) is replaced by REE in zircon while a fraction of Ca by REEs in allanite. The main REE carbonates are bastnasite ($REE(CO_3)F$) and synchysite ($Ca(REE)(CO_3)_2F$). The main phosphates are monazite ($(REE)PO_4$) and xenotime (YPO_4). Fergusonite ($(REE)NbO_4$) and pyrochlore ($(Ca,Na, REE)_2Nb_2O_6(OH,F)$) are the major REE oxide minerals. The gangue minerals consist of silicates (88.2%), oxides (1.8%), calcite, fluorite and apatite (2.2%) and sulfides (0.5%). The estimated reserve is 5 million tons of 0.65% total REE (TREE) at a cut-off of 0.4% (Ucore, 2013).

2.5.2 Froth flotation

Flotation tests were carried out on a presorted REE ore sample by (Ganguli, et al., 2013). The ore sample was a product of the preliminary x-ray sorting process. The contents of LREE and HREE of the ore sample are presented in Table 2.2 and Table 2.3.

Table 2.2 LREE contents in the ore

Elements	La	Ce	Pr	Nd	Sm
Grade (ppm)	2404.0	6331.0	874.0	3614.0	851.0

Table 2.3 HREE contents in the ore

elements	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y
Grade (ppm)	79.0	740.0	123.0	665.0	120.0	286.0	34.0	174.0	17.0	3224.0

The pulp density was set to about 28% w/w. Pulp pH was set using soda ash and hydrochloric acid at the beginning of first-stage flotation. Pulp pH was set to 8.8-8.9. The conditioning time of each flotation reagent was adjusted at 5 minutes. The duration time of skimming froth from the top of the flotation cell was also adjusted at 5 minutes. For test to investigate influence of the grinding time on flotation, first, 0.04 kg/tonne (0.1 lb/t) of Cytec Aero 704 and 0.04 kg/tonne of Cytec Aero Froth 88 were used as the collector and frother, respectively. Flotation with 80-minute grinding compared to that of 60-minute grinding presented better performance for Ce (Figure 2.2). This pattern is similar for LREE such as La, Pr and Nd, and for HREE such as Y, Gd and Dy. A similar flotation test was conducted using Cytec Aero 6493 as the collector. It also was shown that 80-minute grinding was better than with 60-minute grinding for REE mineral flotation. Grinding of 80 minutes was selected for the rest of the flotation tests in the study. When the ore was ground for 80 minutes, particle size of the product was 77% passing 200 mesh (0.075 mm) (Ganguli, et al., 2013).

Cytec Aero 704 and Cytec Aero 6493 were used to check flotation of the REE minerals. The Aero Froth 88 of 0.04 kg/tonne was used in each stage of the flotation tests. First, a dosage of 0.04 kg/tonne of Cytec Aero 704 was applied at each stage. Then, for testing the effect of Cytec Aero 6493, a dosage of 0.04 kg/tonne was used at each stage. Flotation performance of Ce, La, Pr and Nd with the Cytec Aero 6493 as the collector was a little better than flotation performance with Cytec Aero 704 as the collector. This pattern was similar for Y, Gd and Dy. The Ce flotation result is shown in Figure 2.3 (Ganguli, et al., 2013).

Two different dosages of sodium metasilicate were used to examine its effect on REE flotation. Eighty-minute grinding was applied to produce the feed and the pulp pH was set to 8.8-8.9. A dosage of 0.04 kg/tonne of Cytec Aero Froth 88 frother was also applied in each stage. When 0.08 kg/tonne (0.2 lb/t) of sodium metasilicate was used in each stage, the Ce recovery at same mass yield was about 15 percent higher than the recovery when sodium metasilicate was not used. The individual elemental recovery values of La, Pr, and Nd are really identical to those of Ce. Though, when sodium metasilicate dosage was increased to 0.16 kg/tonne in each stage, the REE recovery was lower because of over dose, which also depressed REE silicate minerals. The use of sodium metasilicate assisted Ce flotation considerably with the Cytec Aero 6493 collector. The influence of sodium metasilicate on Y, Gd and Dy flotation was similar. When a dosage of 0.08 kg/tonne of sodium metasilicate was used, higher flotation performance was shown than that of 0 and 0.16 kg/tonne (Ganguli, et al., 2013).

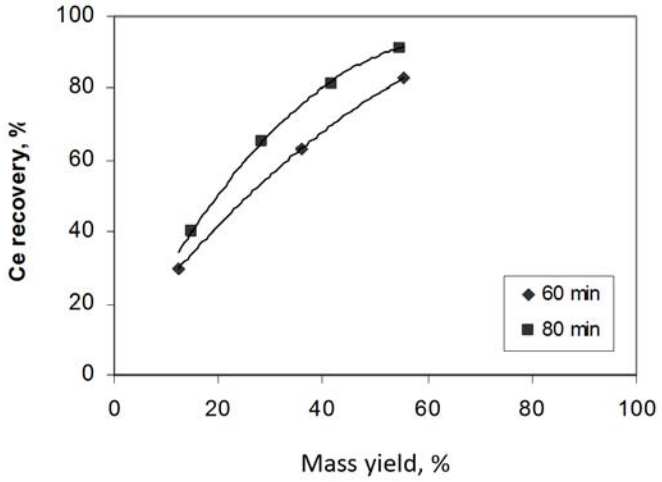

Figure 2.2 Effect of grinding time on Ce flotation (Ganguli, et al., 2013)

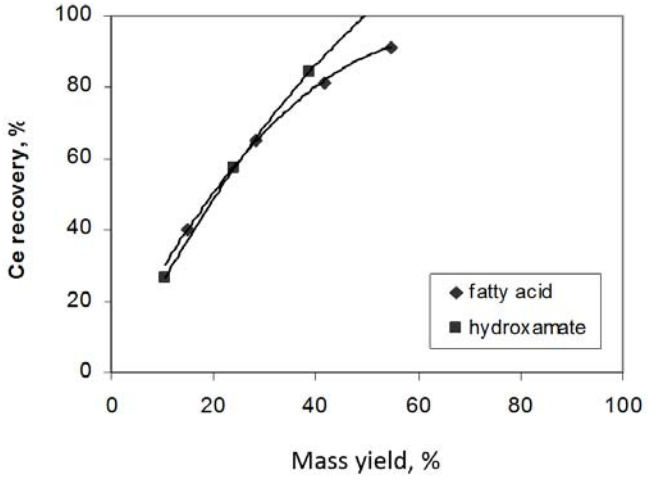

Figure 2.3 Effect of collector on Ce flotation (Ganguli, et al., 2013)

Table 2.4 Flotation test results (Ganguli, et al., 2013)

Product	La Recovery (%)	Ce Recovery (%)	Pr Recovery (%)	Nd Recovery (%)	Y Recovery (%)	Gd Recovery (%)	Dy Recovery (%)
Concentrate	93.9	93.8	93.8	93.5	86.6	90.9	86.8

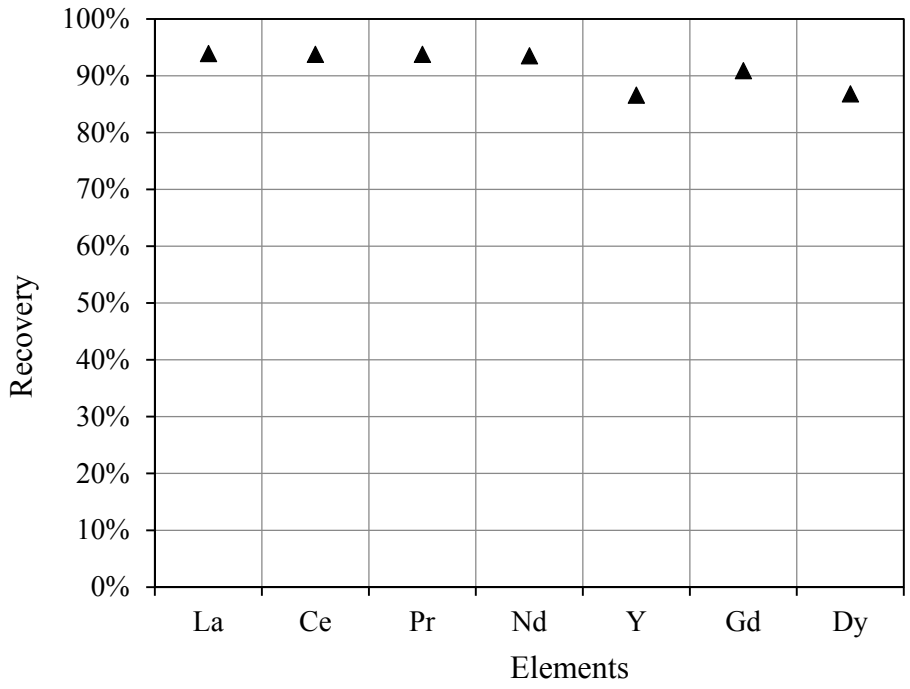

Figure 2.4 REE recoveries of flotation

Ucore (2013) studied the role of collectors (SM 15, FA-1, 727, Linoleic acid, Saponified FA-1, Aero 825), grind sizes, and conditions (caustic, acid, natural pH) in eight froth flotation tests. Feeds were typically 80% passing 38 microns, with a head grade of 1.34% total rare earth oxide (TREO) and Y_2O_3. Achieved recoveries were in 17.1-95.1% range. The tests were exploratory, and not intended to optimize recoveries.

2.5.3. Leaching

2.5.3.1 Direct leaching

Sulfuric acid leaching tests were conducted to determine the effects of time, temperature and the H_2S0_4:ore ratio on yttrium and rare earth extractions of the Bokan Mountain - Dotson Ridge ore (Green, et al., 1995). The ore included 1.26% Ce, 0.10% Dy, 0.14% Gd, 0.54% La, 0.49% Nd, 0.08% Pr, 0.09% Sm and 0.59% Y. Water leaching of the sulfated ore was carried out for 1 hr at 25°C and 20% solids. Results showed that the H_2S0_4:ore ratio was the most significant variable for yttrium, cerium, and lanthanum extraction. Yttrium and lanthanum extraction also showed a dependence on temperature. Sulfuric acid leaching time demonstrated little effect. Thus, the H_2S0_4:ore ratio and the sulfuric acid leaching temperature were tested further. Table 2.5 and Table 2.6 express the effects of increasing the H_2S0_4:ore ratio and the temperature, respectively. Results show that an H_2SO_4:ore ratio of at least 1:1 and a temperature of 250°C are needed for high metal extraction. From this test work, it was determined that a temperature of 250°C, an H_2S0_4:ore ratio of 1:1, and a time of 1 hr were suitable for high yttrium, cerium and lanthanum extraction. (Green G.K. and Harbuck D.D., 1995)

Table 2.5 Effect of increasing H_2SO_4:ore ratio, 300°C, 1 hr (Green G.K. and Harbuck D.D., 1995)

H_2SO_4:ore ratio	Extraction, %		
	Y	Ce	La
0.25:1	59	74	64
0.5:1	51	86	76
0.75:1	52	86	76
1:1	76	100	88
1.25:1	75	99	86
1.5:1	78	100	88

Table 2.6 Effect of increasing sulfation temperature, H_2SO_4:ore ratio of 1:1, 1 hr (Green G.K. and Harbuck D.D., 1995)

Temperature, °C	Extraction, %		
	Y	Ce	La
100	42	54	45
150	68	94	77
200	74	94	81
250	75	100	87
300	75	100	88

A sample of the Zone 25-Selected material was tested. This sample had a particle size of 191 microns (80% passing 191 microns). The grade of the sample was higher than the average grade of the deposit. Contents of LREE and HREE+Y were 28,200-28,890 ppm and 12,910-13,160 ppm, respectively. The sample was leached by 30% (v/v) nitric acid in two stages at 90°C. Solids percentage was 20%. First, the concentrate was leached in lean acid to produce the process leach solution, having the leached REE after beneficiation. Second, the concentrate was subjected to new nitric acid in the second stage of the leaching. Recoveries of LREE and HREE were 93-97% and 75-83%, respectively (Ucore, 2013).

Hydrochloric acid (HCl) leaching with 20% HCl was better than leaching in 10% HCl (Ganguli, et al., 2014). It was 2-stage leaching. The duration time was 2 hours in each stage. The leaching was conducted at 90°C in each stage. In the leaching with 20% HCl in the 1st and 2nd stages, which presented the best recoveries of REE, the individual elemental recoveries were 92.1% Ce, 92.1% La, 92.0% Pr, 92.1% Nd, 70.5% Y, 85.2% Gd and 76.7% Dy (Table 2.7). La recoveries were 92.1% and 87.3% for 20% HCl and 10% HCl, respectively. This pattern was similar for Ce, Pr and Nd. For the Gd, Dy and Y; 20% HCl leaching also was better than leaching in 10% HCl. Typically, their individual elemental recovery values were lower than those of LREE (Ganguli, et al., 2014).

Table 2.7 REE recoveries of hydrochloric acid two-stage leaching of the REE ore: 2 hours in each stage (Ganguli, et al., 2014)

	Acid concentrations	Recovery (%)						
		Ce	La	Pr	Nd	Y	Gd	Dy
1	20% HCl in 1st stage and 20% HCl in 2nd stage	92.1	92.1	92.0	92.1	70.5	85.2	76.7
2	10% HCl in 1st stage and 10% HCl in 2nd stage	86.6	87.3	86.6	86.5	56.3	75.0	60.9

2.5.3.2 Leach after flotation

The leach after flotation or Flotation/Leach process consisted of 2 stages (Ganguli, et al., 2014). The first stage was the flotation test. The second stage was the leaching test conducted on the first concentrate of the flotation. For the leaching of the concentrate 1 of flotation, La recovery of the leaching with 20% HCl in the 1st and 2nd stages was greater than that of leaching with 10% HCl in both stages. La recoveries were 90.6% and 96.2% for the 2nd and 1st options of the HCl concentrations, respectively (Table 2.8). Ce, Pr and Nd individual elemental recovery values were similar to those of La (Table 2.8). Gd, Dy and Y individual elemental recoveries of the leaching with 20% HCl in the 1st and 2nd stages were also higher than those of leaching with 10% and 10% HCl. Generally, individual elemental recoveries were lower than those of the LREE (Table 2.8) (Ganguli, et al., 2014).

Table 2.8 REE recoveries of hydrochloric acid two-stage leaching of concentrate 1 of flotation: 2 hours in each stage (Ganguli, et al., 2014)

	Acid concentrations	Recovery (%)						
		Ce	La	Pr	Nd	Y	Gd	Dy
1	20% HCl in 1st stage and 20% HCl in 2nd stage	96.5	96.2	96.7	96.6	66.5	89.5	73.5
2	10% HCl in 1st stage and 10% HCl in 2nd stage	90.8	90.6	91.2	91.3	55.7	83.3	64.0

Flotation/Leach net recoveries of La, Ce, Pr and Nd were 90.3%, 90.4%, 90.6% and 90.3%, respectively. Gd, Dy and Y net recoveries in the Flotation/Leach process were 81.2%, 63.5% and 57.2% (Table 2.9, Figure 2.5). The Flotation/Leach net recovery was lower than flotation independent recovery and leach independent recovery because the leaching was carried out on the concentrate of the flotation.

Table 2.9 Net recovery of the combined Flotation and Leach process

Elements	La	Ce	Pr	Nd	Y	Gd	Dy
Recovery (%)	90.3	90.4	90.6	90.3	57.2	81.2	63.5

The Flotation/Leach test is compared next with the direct leach test. Figure 2.6 shows that all recoveries of La, Ce, Pr, Nd, Gd, Dy and Y in the Flotation/Leach process are lower than recoveries of the same elements in the direct leach process. For example, La recoveries were 90.3% in the Flotation/Leach process and 92.1% in the direct leach process.

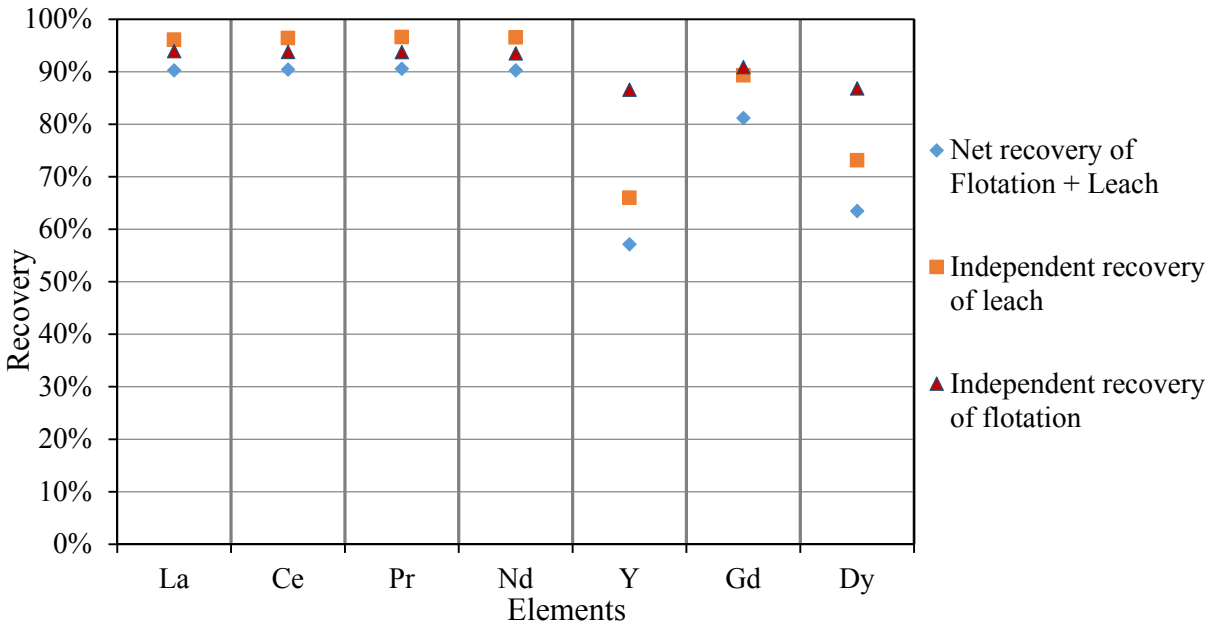

Figure 2.5 Recovery of Flotation/Leach net recovery, independent recoveries of leach and flotation

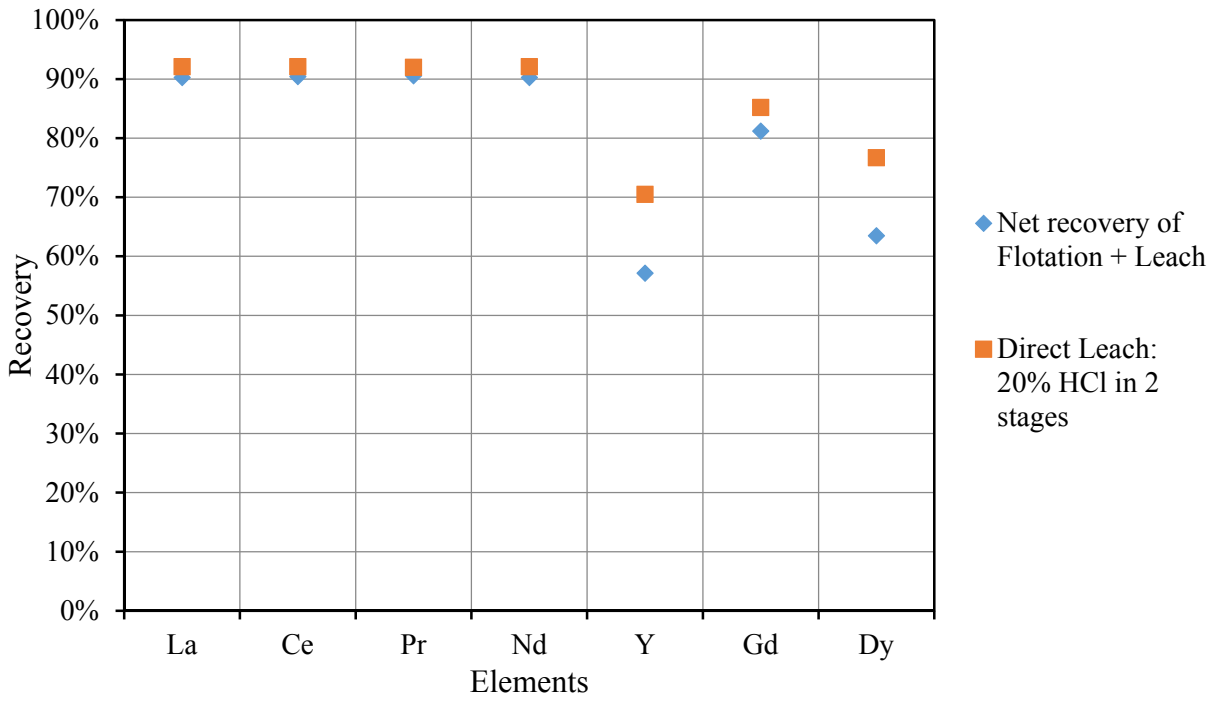

Figure 2.6 Comparison between Flotation/Leach and Direct leach

Chapter 3 Methods and Materials

3.1 Materials

An ore sample of the Bokan Mountain – Dotson Ridge rare earth elements (REE) deposit was selected for this beneficiation study. The sample was delivered to the mineral processing laboratory of the Mineral Industry Research Laboratory (MIRL). The received sample was a +1/4 inch composite of a concentrate presorted by an x-ray sorting process from its raw ore. The sample, about 25 kg, was dried in an oven (Figure 3.2) and crushed. The crushed sample was split.

The sample was stage crushed to -6 mesh (3.36 mm) using a jaw crusher and then screened in a screen with an aperture of 6 mesh to prevent it from being over crushed. The over screen material was then reduced to -6 mesh in a double-roll crusher (Figure 3.1).

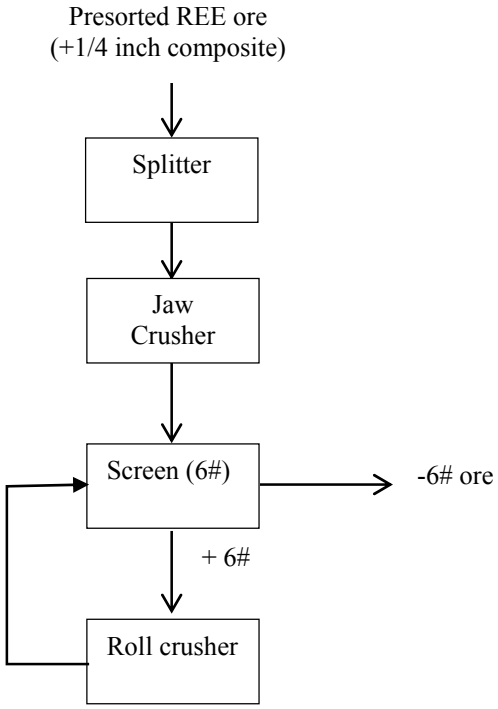

Figure 3.1 Crushing the x-ray sorted ore

a

b

c

Figure 3.2 Drying the raw ore received from the Bokan deposit: a) the ore before drying, b) weighing the ore, c) drying the ore in an oven

The -6 mesh ore was mixed well in a bar riffle splitter, then split to obtain many 700 gram samples which were ready for the metallurgical tests. The samples were sealed in plastic bags to prevent further oxidation.

A split of the crushed sample was sent to a commercial laboratory for REE analyses. LREE and HREE contents in the sample were analyzed by lithium metaborate fusion followed by ICP-MS analysis. The LREE and HREE contents in the sample are indicated in Table 3.1. The grade of TREEs is approximate 2.0%.

Table 3.1 Grades of the REE in the feed ore

Elements	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y
Grade (ppm)	79	740	123	665	120	286	34	174	17	3224
elements	La	Ce	Pr	Nd	Sm					
grade (ppm)	2404	6331	874	3614	851					

3.2 Gravity separation tests

For the gravity concentration tests, the split representative samples of the presorted REE ore were processed two different ways in order to better understand liberation. Feed of the gravity separation test was 700 g of the crushed ore. The panning technique was used as a gravity concentration test. The two test protocols allow for the staged liberation of the rare earth minerals and subsequent attempts to recover them by gravity concentration.

In one approach, first, the -6 mesh ore was gravity concentrated (Figure 3.3). Then its tailings were serially ground 3 times to determine the amount of liberation occurring at different levels of grinding. A sample of concentrate was collected and analyzed after each instance of grinding. The process in Figure 3.3 shows three levels of 5-minute rod mill grinding. This approach can be varied by increasing or decreasing the duration and level of grinding. For example, the duration of grinding can be decreased to 3 minutes and the level of grinding can be increased to four levels. Figure 3.3 shows the flowsheet that was followed for Test 1, while Figure 3.4 shows the flowsheet that was followed for Test 3, a process similar to that in Figure 3.3, but fine-tuned after studying

the results of Test 1. In Test 3, there are 3 levels of grinding. The first grinding level has a duration of 5 min, the second has 4 min and the third has 3 min. Gravity concentration was followed by each grinding level. A sample of concentrate was collected and analyzed after each instance of grinding.

In the second approach (Figure 3.5 and Figure 3.6), the intent is to increase the grade through regrinding. Figure 3.5 and Figure 3.6 show the flowsheet followed for Test 2. The processes followed for Test 4 are shown in Figure 3.7 and Figure 3.8. In Test 2, a sample was serially ground several times and a sample of its concentrate was collected after each instance of grinding (Figure 3.5). It had three levels of 4-minute rod mill grinding in a general stage. Then concentrates 1, 2, 3 and 4 were combined and gravity concentrated (Figure 3.6). A concentrate was taken up. Its tailings were reground for 30 seconds and gravity concentrated. The last concentrate was added to the previous concentrate taken before regrinding. The combined concentrate was the final concentrate. The final concentrate, middlings, and tailings were collected and analyzed. In test 4, first, the -6 mesh ore was gravity concentrated (Figure 3.7). Then its tailings were ground serially. There were 5 levels of grinding in a general stage. The first 3 grinding levels had 3-minute rod mill grinding and the other 2 levels had 2-minute grinding. Gravity concentration was performed after each grinding level. Then concentrates C_a, C_b, C_c and C_d were combined and reground (Figure 3.8). After regrinding, reground concentrate was added to concentrates C_e and C_f. The last combined concentrate was gravity concentrated three times producing concentrates C_1, C_2 and C_3, and middlings (Figure 3.8). The concentrates, middlings and final tailings were collected and analyzed (Figure 3.7 and Figure 3.8).

This procedure employs panning as the gravity concentration test because it allows accuracy and flexibility in working with small sample masses and also provides an easy means to control concentrate mass yield (%) or concentration ratio.

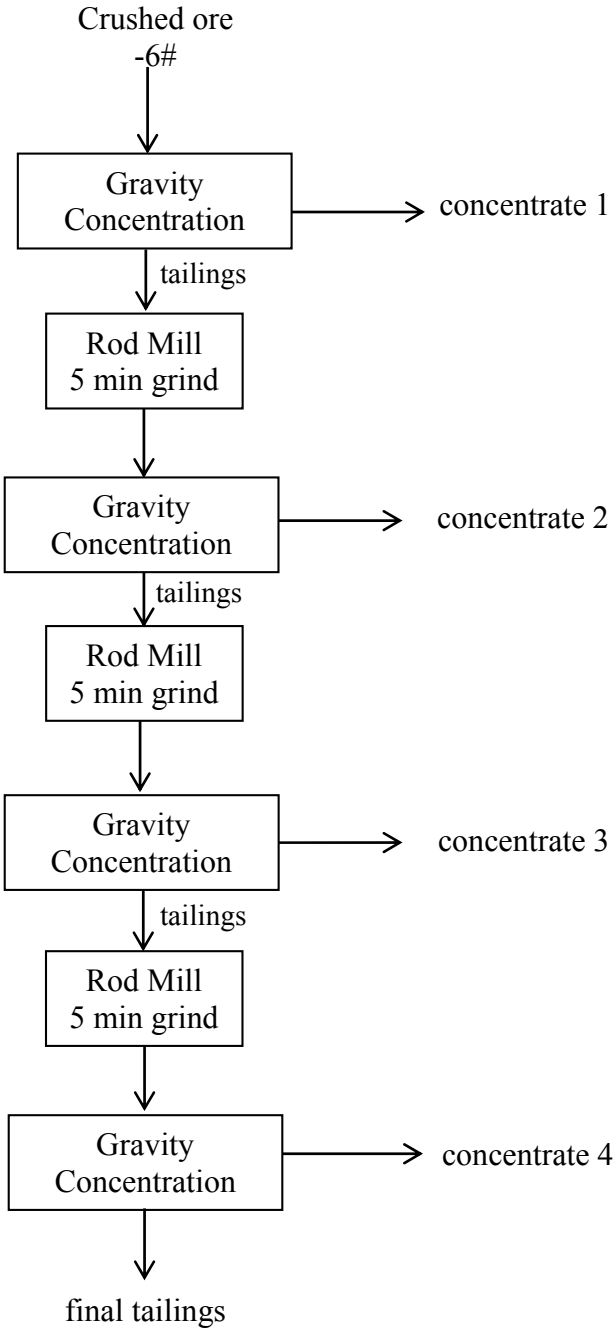

Figure 3.3 Flowsheet of gravity separation test 1

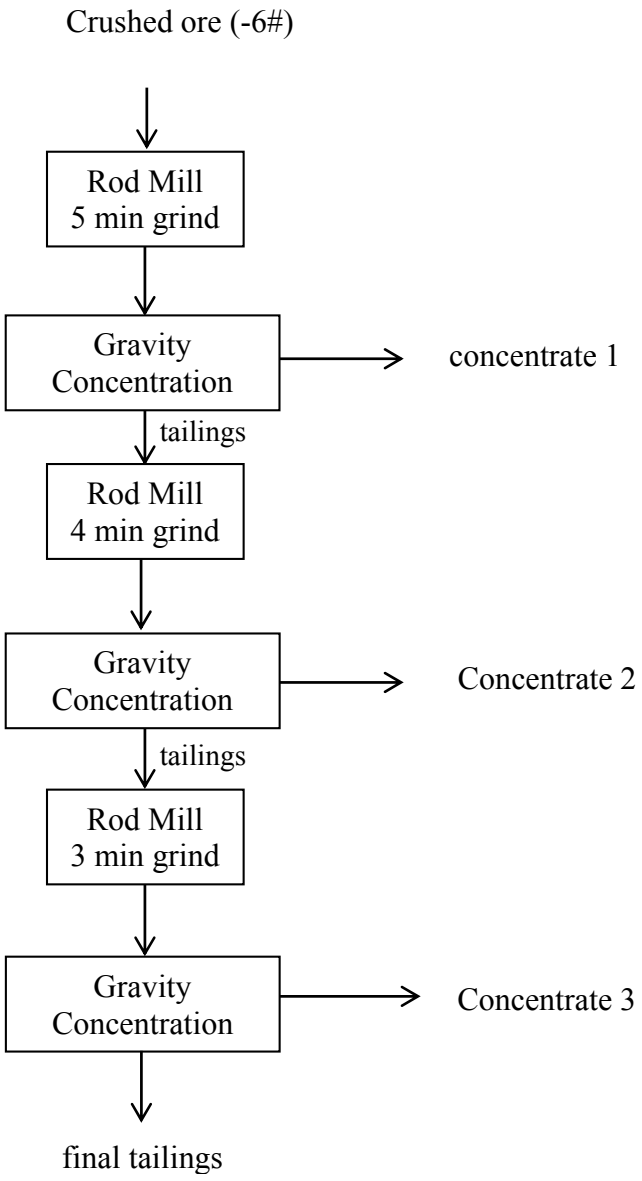

Figure 3.4 Flowsheet of gravity separation test 3

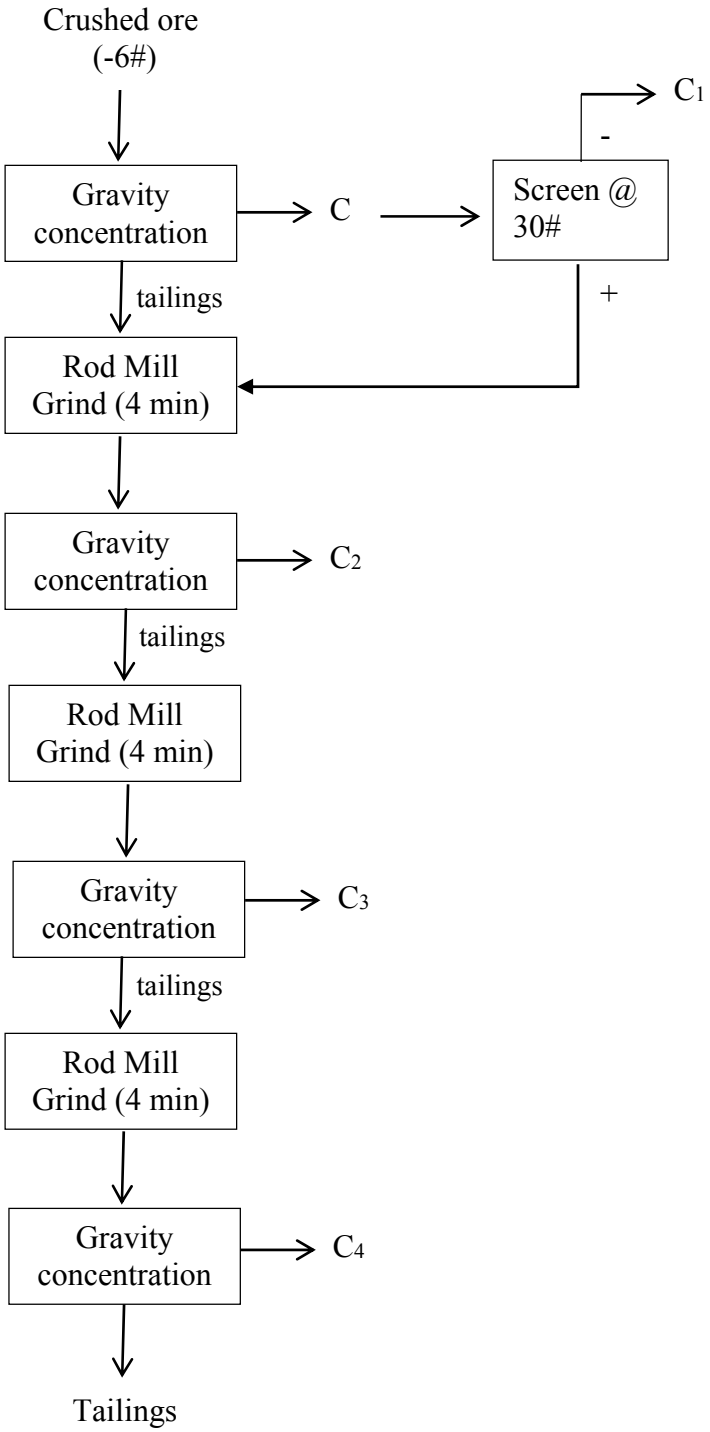

Figure 3.5 Flowsheet of gravity separation test 2 (continued in Figure 3.6)

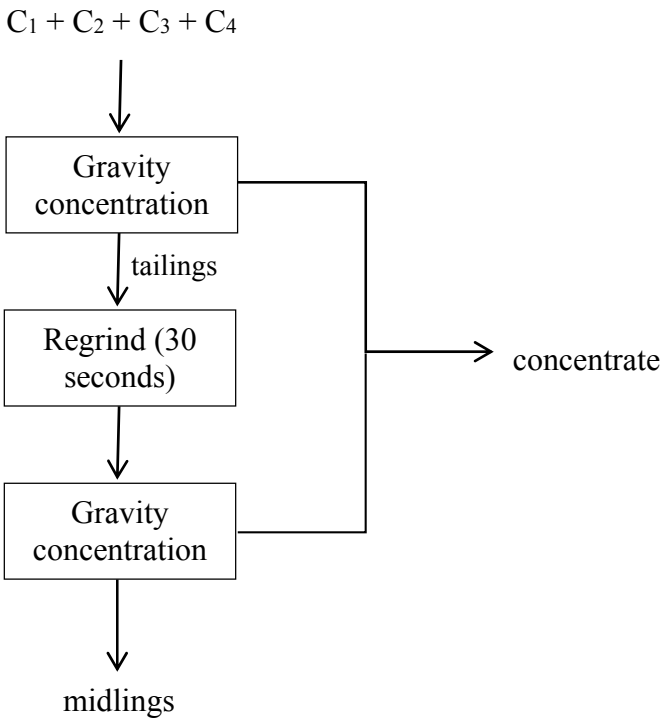

Figure 3.6 Partial flowsheet for the test 2

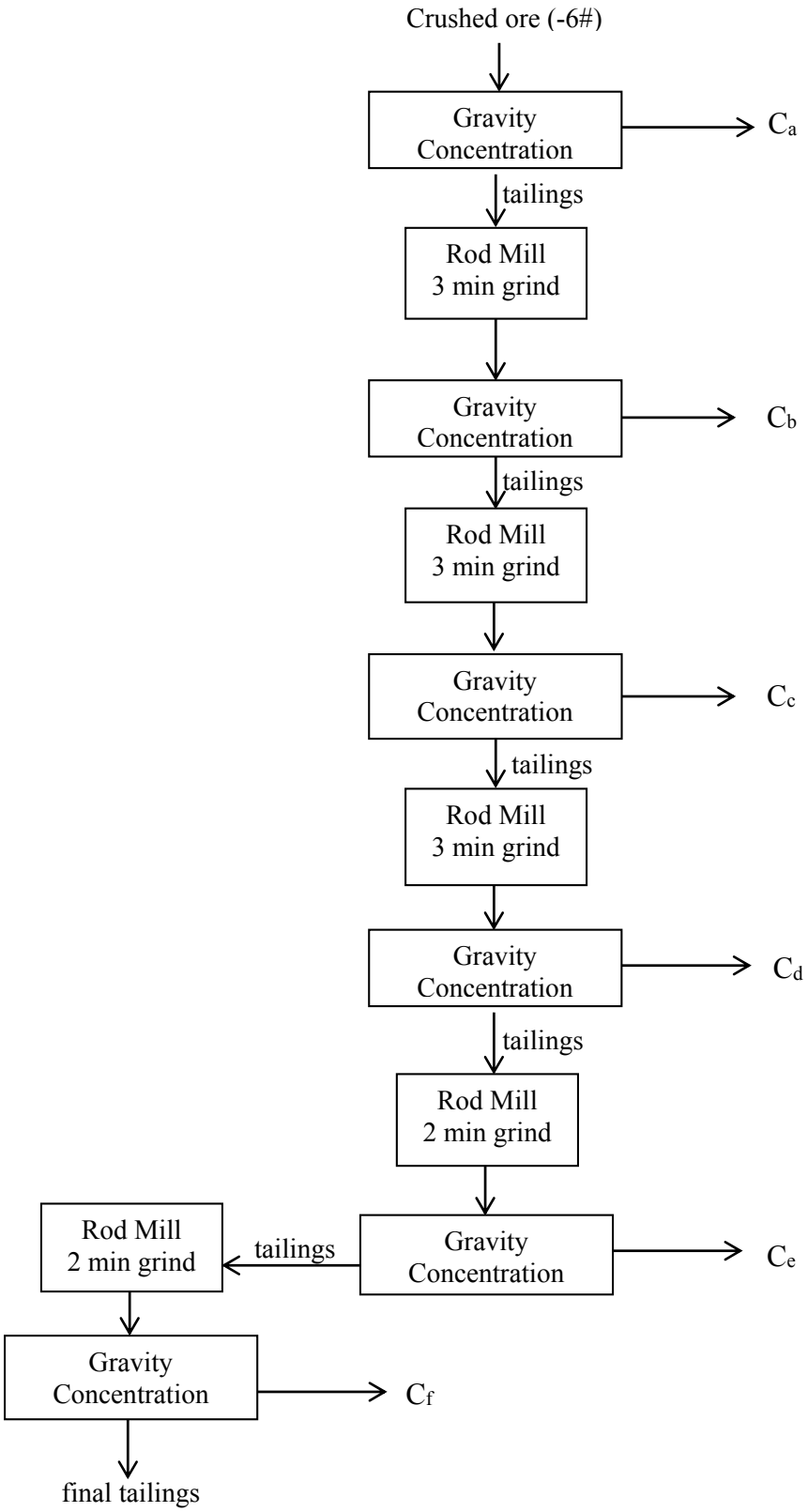

Figure 3.7 Flowsheet of gravity separation test 4 (continued in Figure 3.8)

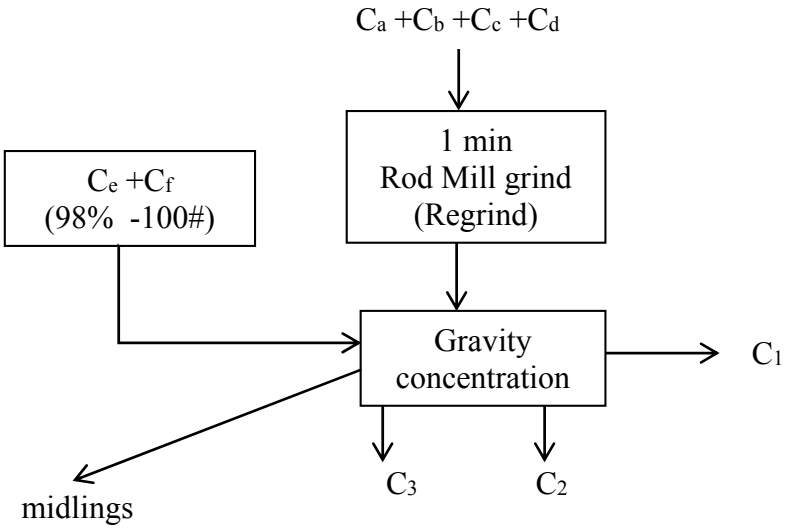

Figure 3.8 Partial flowsheet for the test 4

3.3 Wet high intensity magnetic separation tests

The magnetic separation tests were conducted on the tailings of the gravity separation tests in order to recover REE minerals. The flowsheet for the magnetic separation tests is shown in Figure 3.9. A Carpco Wet High Intensity Magnetic Separator (WHIMS) was used. The magnetic separation test consisted of 5 stages based on the WHIMS's amperage sets of 0.2A, 0.4A, 0.6A, 0.8A and 1.0A.

In the first stage, 50 g of tailings of a gravity separation test were separated by WHIMS operated at 0.2A. The solid percentage of the feed was 20%. The concentrate was washed out by 200 ml water. It was named Concentrate 1. The tailings and washed water were mixed and allowed to settle. After it had settled, the liquid phase was transferred to a vacuum filter to separate water and to monitor solid discharge in the liquid phase.

Settled tailings were then prepared as feed for the second stage of the magnetic separation. Water was added to the settled tailings to achieve 20% solids in the feed. It was separated by WHIMS operated at 0.4A. Concentrate #2 was washed out by 200 ml water. Tailings and washed water were mixed and allowed to settle. After they settled, the liquid phase was transferred to a vacuum filter to separate water.

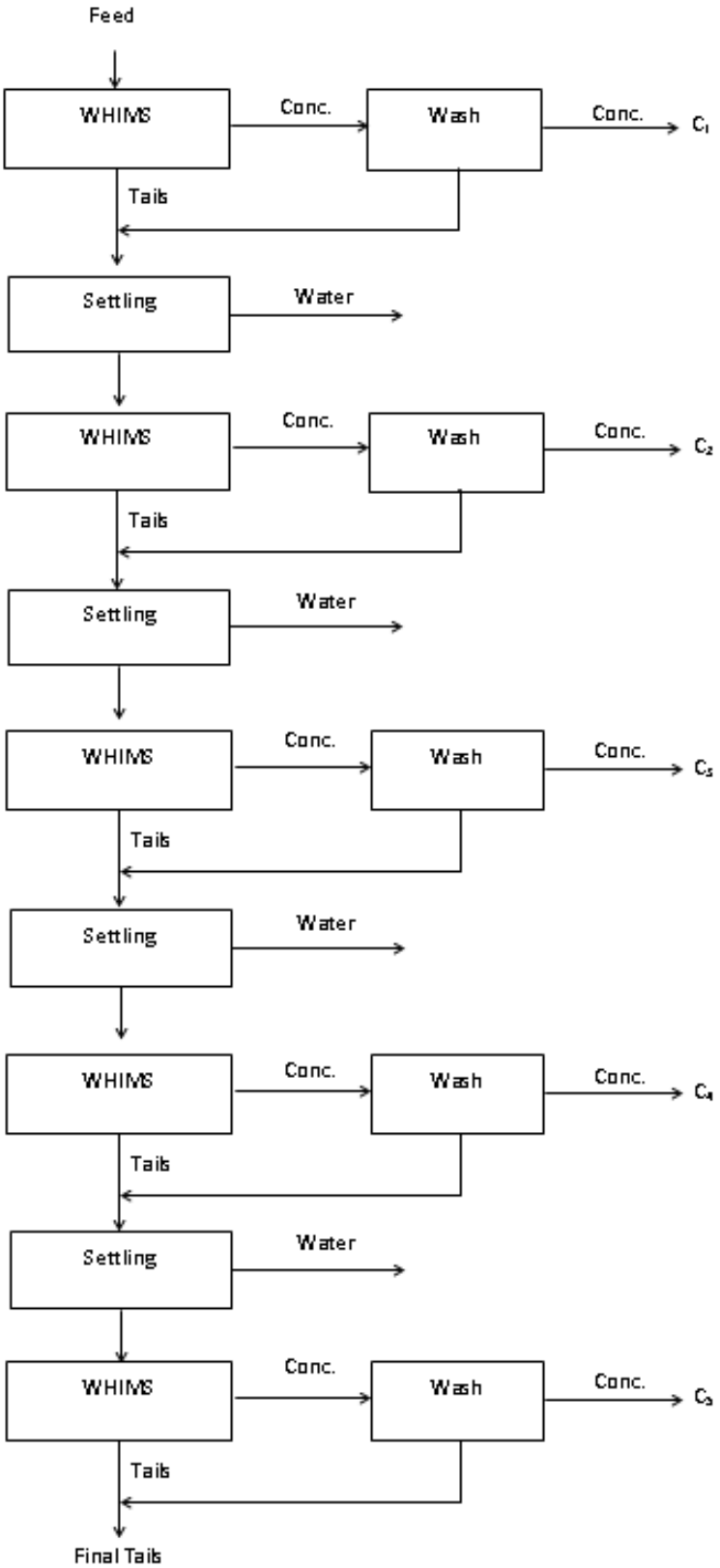

Figure 3.9 Flowsheet of wet high intensity magnetic separation tests

In the 3rd stage, settled tailings from the second stage were separated in WHIMS in 0.6A. The solid percentage of the feed was once again brought to 20%. Concentrate #3 was washed out by 200ml water. Tailings and washed water were mixed and allowed to settle. After it had settled, the liquid phase was transferred to a vacuum filter.

In the 4th stage, settled tailings from the previous stage were separated in WHIMS in 0.8A. The solid percentage of the feed was 20%. Concentrate #4 was washed out by 200 ml water. Tailings and washed water were mixed and allowed to settle. After it had settled, the liquid phase was transferred to a vacuum filter.

In the 5th stage, settled tailings from the previous stage were separated in WHIMS in 1.0A. The solid percentage of the feed was adjusted to 20%. Concentrate #5 was washed out by 200 ml water. Tailings and washed water were mixed and allowed to settle. After it had settled, the liquid phase was transferred to a vacuum filter. Tailings of the 5th stage were the final tailings.

Settled tailings of the 5th stage were transferred to a vacuum filter to separate water. All 5 concentrates were filtered independently. All the concentrates and final tailings were dried in an oven at 93°C. And then mass of each product of the test was measured. After that, representative samples taken from each product were sent for assaying.

3.4 Froth flotation tests

Cytec Aero 6493 collector and sodium metasilicate depressant were selected for the flotation. At each stage of the flotation test, 0.05 kg/tonne of Cytec Aero 6493, 0.1 kg/tonne of sodium metasilicate and 0.05 kg/tonne of aliphatic alcohol frother (Cytec Aero Froth 88) were applied. The froth flotation test was carried out on the split representative sample of the presorted REE ore. Before the flotation test, the feed ore was ground in a 5.7 inch inner diameter steel ball mill. For the grinding, the solid percentage of the slurry in the mill was 50% by weight. The grinding speed was adjusted to 83 rpm. Grinding time was 65 min. Particle size of the 65-minutes grinding product was 82% passing 200 mesh (0.075 mm).

A 2-liter Denver conventional flotation cell was selected for the test work. The flowsheet of the flotation tests is shown in Figure 3.10. In each flotation test, the initial solid percentage of pulp was adjusted to approximately 28% by weight. Slurry pH was adjusted to 8.8-8.9 by adding soda ash and/or hydrochloric acid prior to the first-stage flotation. The conditioning time for each addition of flotation reagents was set at 5 minutes. The duration of skimming mineralized froth was set at 4 minutes. REE grades of flotation concentrates and tailings were analyzed by lithium metaborate fusion followed by ICP-MS analyses in a commercial assaying laboratory.

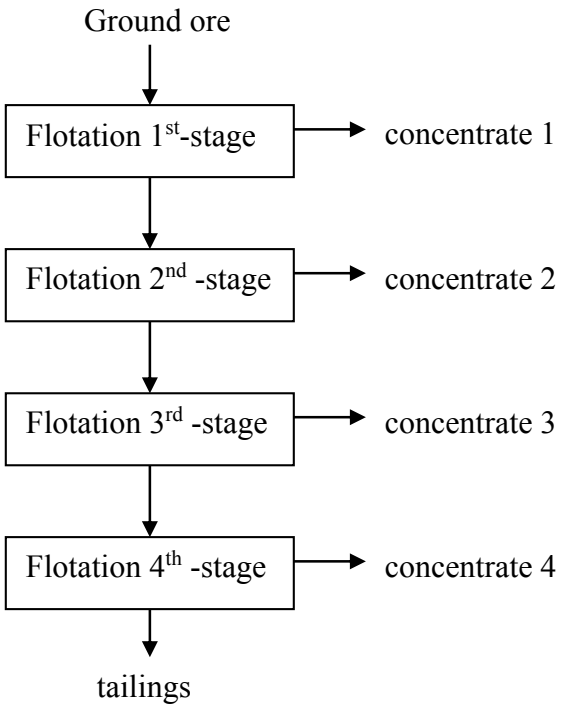

Figure 3.10 Flowsheet of the flotation test

3.5 Leaching tests

Leaching was conducted on two different feeds. In one, termed direct leaching, the feed was the x-ray sorted ore. In the second, termed "leach after flotation," the feed was the concentrate of the flotation test. The leach method was identical in both cases.

3.5.1 Direct leaching

The direct leaching tests were conducted on the split representative samples of the presorted REE ore. Its fineness was 77% passing 200 mesh (0.075 mm). A 100 ml beaker was selected as a reactor for the test. The leaching tests were conducted at 90°C. A Thermolyne Cimarec2 hot plate with magnetic stirrer was used to heat and stir the slurry in the beaker. It was a 2-stage leaching (Figure 3.11). The duration time of each stage was 2 hours. The feed of the leaching test was 5 g of the ground ore. The solid percentage of the leach slurry was 20% w/w.

In the first stage of the leaching, 5.00 g of the ground REE ore sample was placed in a 100 ml beaker and 20.0 g (18.2 mL) of 20% hydrochloric acid (HCl) was added to it (Table 3.2). The slurry was heated to 90°C on the hot plate. Agitation was started and the leaching continued for 2 hours. Then the pregnant leach solution (PLS) was decanted to a 100 ml volumetric flask. Residue of the first stage of the leaching was rinsed out two times by 20 ml 5% of HCl in the reactor. The rinse solutions were decanted to the same volumetric flask.

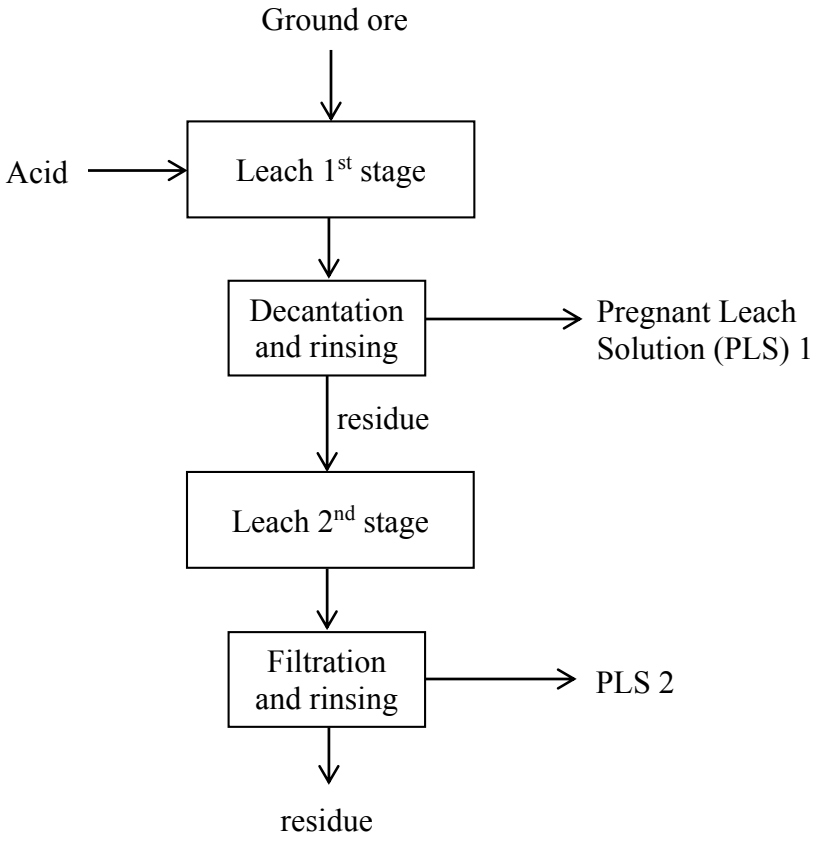

Figure 3.11 Flowsheet of leaching tests

The remaining leaching residue was leached in the second leaching stage. 20.0 g of 20% HCl was added to the first leaching stage residue (Table 3.2). The slurry also was heated to 90°C on the hot plate. Agitation was started and the leaching again lasted for 2 hours. Then the PLS was filtered to 100 ml volumetric flask. Residue of the second stage of the leaching was rinsed out two times by 20 ml 5% of the selected acid. The filtered cake was dried in an oven at 90°C. The cake samples were taken for analysis.

Table 3.2 Hydrochloric acid concentrations in 2 stages in the leach tests

Stages	Concentration of HCl (%)
1st stage	20
2nd stage	20

3.5.2 Leach after flotation

For the leach after flotation tests, the feed was the concentrate of the first stage of the flotation test (Figure 3.10). The solution of 20% HCl was applied in each stage of these tests. The 2-stage leaching procedure (Figure 3.11) was followed. The duration time was 2 hours for each stage. Chosen factors such as acid concentration, solid percentage of slurry and temperature of heating were the same as for the direct leaching tests. The filtered cake was dried in an oven at 90°C. The cake samples were taken for analysis.

3.6 Electron microprobe analysis of REE mineralogy

The sample for electron microprobe analysis was prepared by polishing in a resin block. A specific 1-inch cylinder, which is a frame of the resin block, was held on a transparent thermo-resistant sheet by gluing its edge at one end. The sheet was kept on a thermo-plate with a temperature of 130°C. About 2 g of the material was added and a glue mixture was poured into the cylinder. After that, this mixture was kept at 130°C for about 20 min until it was solid (Figure 3.12). After the resin block was ready, it was polished using disc grinders, abrasive powders and polishing discs (Figure 3.13). Abrasive powders were 400 mesh, 600 mesh, 1000 mesh and 3 microns. Polishing discs were 1 to 4 microns. Then the resin block was carbon-coated in 250Å (Figure 3.14). The

sample was analyzed on a JEOL JXA-8530F Electron Microprobe analyzer equipped with a Thermo Scientific UltraDry silicon drift energy dispersive detector controlled by Thermo System Seven NSS software version 3.2. The accelerating voltage was 30 kV. The beam current was 100 nA. Magnification was set to 1000x. The acquiring time was 1 sec. Minimum counts were 5,000. Three regions were selected and the Feature Sizing option of the NSS software was used to collect spectra (1 second each) of particles that showed high brightness in the backscattered electron image (i.e., particles with high average atomic number). Spectra were processed with the NSS standardless quantification software for concentrations of Na, Mg, Al, Si, P, S, K, Ca, Ti, V, Mn, Fe, Co, Zn, Y, Zr, Nb, Cd, Ba, La, Ce, Pr, Nd, Pm, Sm, Eu, Gd, Tb, Dy, Ho, Er, Tm, Yb, Lu, Hf , Ta, Au, Tl, Pb, Th and U.

Filter of Feature sizing was chosen on greater than 1 μm^2 of particles' area. Thresholds were set at 60,000-65,535 of brightness of the electron images. This means that high atomic number elements were selected from the analyzed regions, which were related to the threshold set. Such a threshold is one kind of filtration. The complete range of the brightness of the electron images was from 0 to 65,535.

Figure 3.12 The resin block on a thermo-plate

Disc grinder Abrasive powders

Figure 3.13 Disc grinder and abrasive powder

Figure 3.14 Carbon coated 1-inch diameter resin block

Chapter 4 Results and Discussion

4.1 Gravity separation

The LREE and HREE contents in the feed ore are presented in Appendix E (also shown in Chapter 3). The grade of TREE was approximately 2.0%. Gravity Test No. 1 shows that for a combined concentrate mass yield of 10.3%, light rare earth element (LREE) recovery ranged from 34.9-38.6%, heavy rare earth element + yttrium (HREE) ranged from 32.7-34.3%, and uranium and thorium recoveries were 29.2% and 29.8%, respectively. Test No. 2 demonstrates how the combined concentrates from Test No. 1 might be further upgraded after regrinding, yielding a concentrate of only 3.4%. LREE recovery ranged from 19.5-23.6% and HREE ranged from 17.9-19.6%. The concentrate had an enriched grade on the order of 6-7 times that of the feed material. REE contents in the concentrate for Test No. 1 and Test No. 2 are shown in Appendices E and F, respectively.

Test No. 3 shows that for a combined concentrate mass yield of 25.5%, LREE recovery ranged from 49.6-52.8%, HREE ranged from 46.3-48.8%, and uranium and thorium recoveries were both 42.1%. Test No. 4 demonstrates how the combined concentrates from Test No. 3 might be further upgraded after regrinding, for various concentrate mass yields (3.9%, 7.1% and 10%). Grade enrichment is inversely proportional to recovery. Figure 4.1 summarizes the relationship between concentrate mass yield (%) and recovery for lanthanum. The linear trend is similar for other LREE and for HREE, but recoveries vary slightly for other elements. REE contents in the concentrate for the Test No. 3 and Test No. 4 are shown in Appendices G and H, respectively.

Table 4.1 summarizes the combined gravity recovery and liberation characteristics of the ore sample for the two main levels of mass yield investigated and at various levels of comminution (P95). It is apparent in Table 4.1 that the ore sample has a very significant percentage of REE mineralogy that is liberated at coarse comminution levels and which is available for gravity recovery.

Roughly 50% of the REE mineralogy in the ore samples seems amenable to gravity concentration at a 25% mass yield (Figure 4.1). Regrinding rougher gravity concentrates and re-concentration increases concentrate grade, but at the expense of recovery in the cleaner circuit.

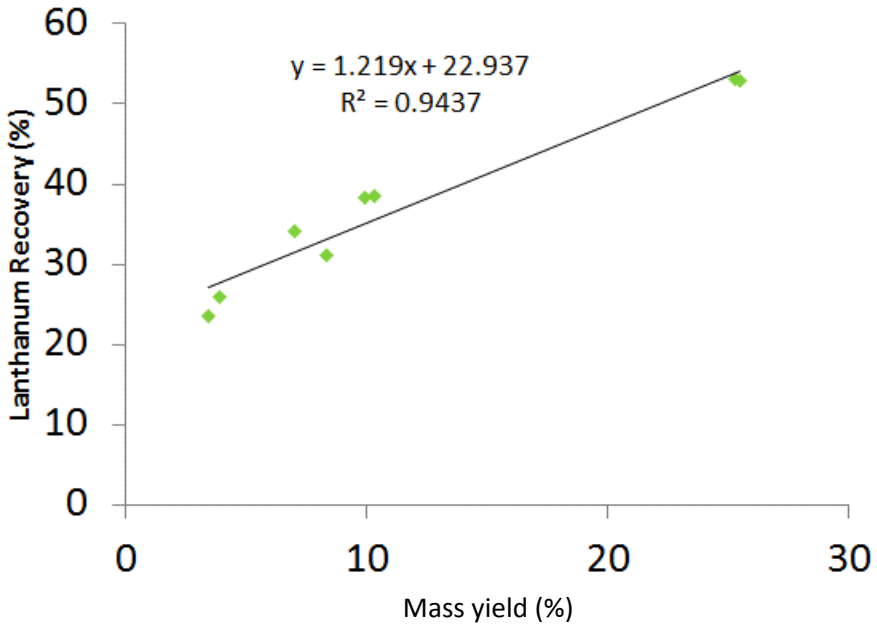

Figure 4.1 The relationship between gravity concentration mass yield and recovery for lanthanum

Table 4.1 REE recovery of the gravity separation tests, which is correlated to particle size and mass yield

	Yield (%) / ~ P95	10	25
1	6 mesh	9.0%	N/A
2	40 mesh	12.6%	30.4%
3	80 mesh	11.5%	14.7%
4	100 mesh	5.5%	7.6%

4.2 Wet high intensity magnetic separation

A wet high intensity magnetic separation (WHIMS) test was conducted on tailings of the gravity separation test. REE contents of the feed of the WHIMS test are shown in Appendix I. Figure 4.2 and Figure 4.3 show the individual elemental recovery values of the wet high intensity magnetic separation test for the LREE and HREE. The recoveries are a little bit higher for the HREE. It appeared that the HREE were separated from the tailings of the gravity separation with recovery ranging from 56.3-63.1% at 37% mass yield of combined concentrate (Figure 4.3).

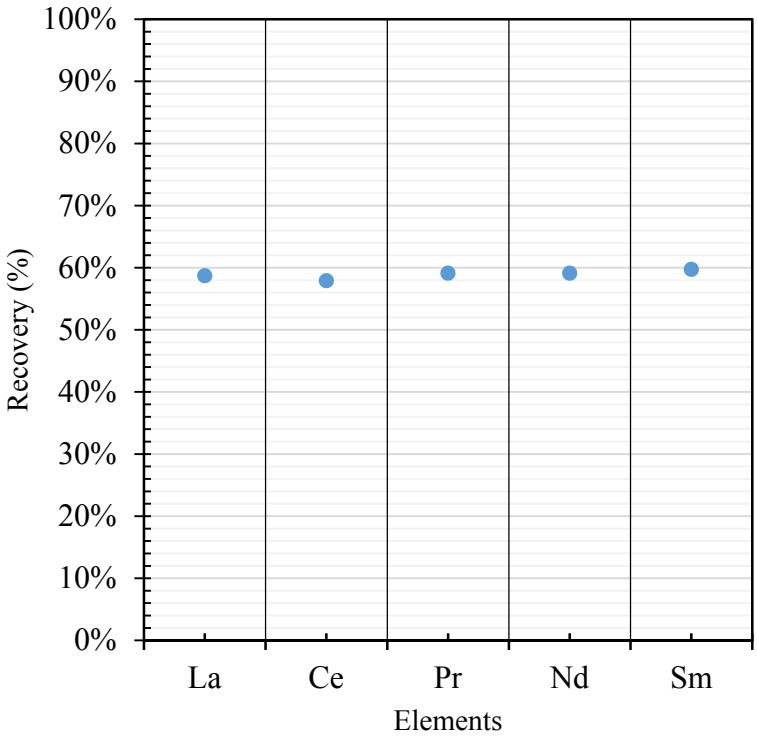

Figure 4.2 LREE recoveries of WHIMS test

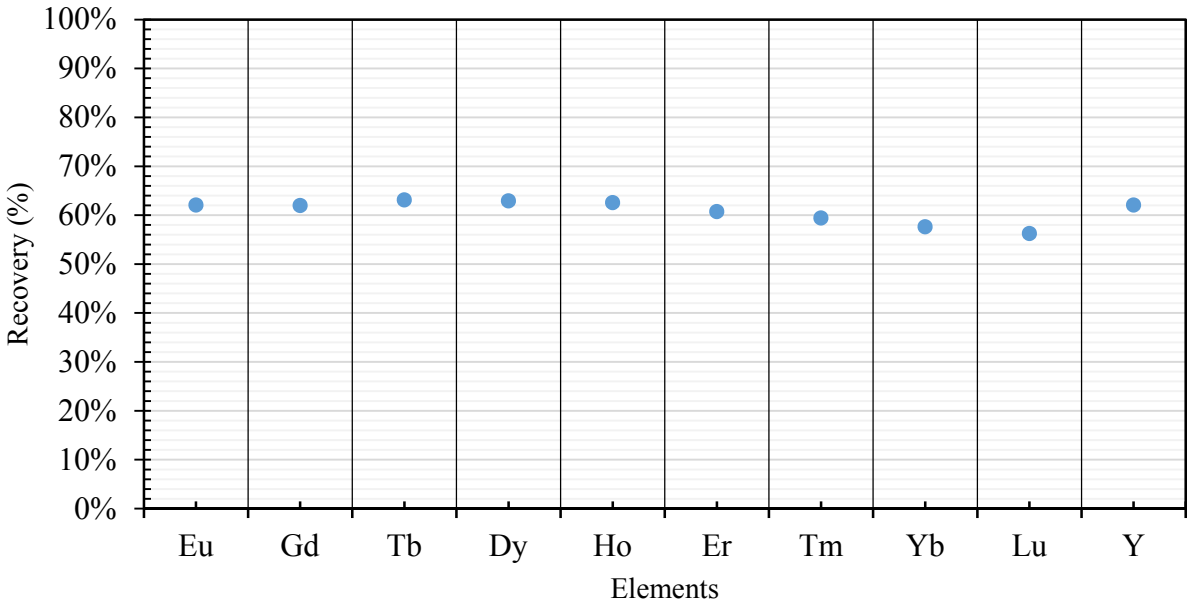

Figure 4.3 HREE recoveries of WHIMS test

For recovery of the light rare earth elements, it was observed that LREE were separated from the tailings of the gravity separation with recovery ranging from 57.9-59.1% at 37% mass yield of combined concentrate (Figure 4.2). REE contents in the concentrate of the WHIMS test are presented in Appendix I. For the Gravity/Magnetic separation process, its net individual elemental recovery values of the LREE and the HREE were in the range of 79.6-80.5% and 76.5-80.9%, respectively. The combined mass yield of the Gravity/Magnetic separation process was 53%.

4.3 Froth flotation

The flotation test was carried out on the representative split of the presorted REE ore sample (see Chapter 3). Contents of LREE and HREE in the feed ore are presented in Appendix J.

Results showed individual elemental recovery values in the range of 44.6-50.4% for the LREE at a mass yield of 23% (Figure 4.4). The HREE individual elemental recovery values ranged from 27.9-44.5% (Figure 4.5). REE contents in the concentrate of the flotation test are shown in Appendix J. The achieved recoveries were significantly lower than those achieved by Ganguli et al (2013). It is possible that the particles were coarser, resulting in lower recoveries.

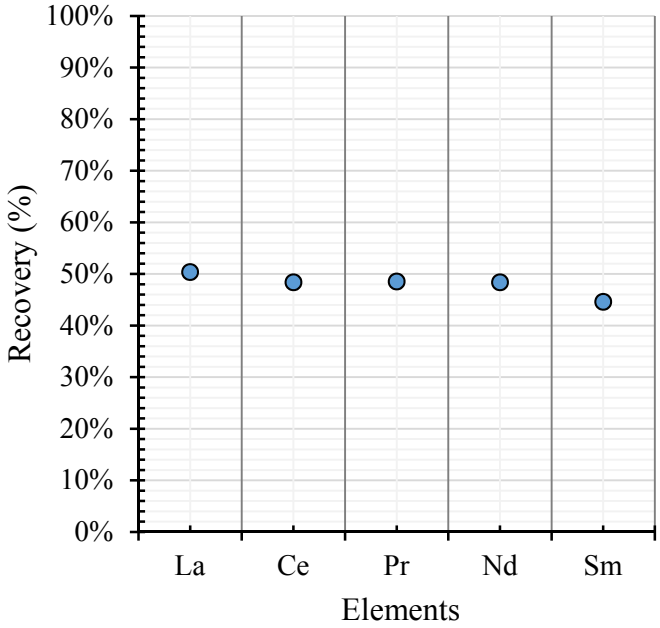

Figure 4.4 Recoveries of LREE as a result of flotation

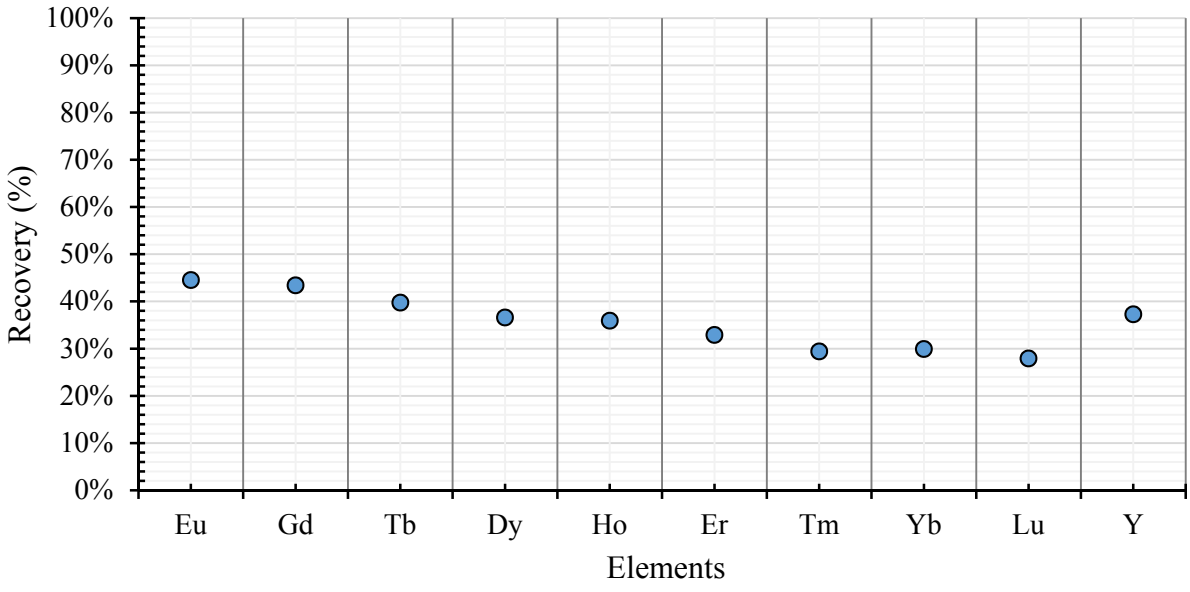

Figure 4.5 Recoveries of HREE as a result of flotation

4.4 Leaching

4.4.1 Direct leaching

Direct leaching tests were conducted on the composite ground ore. The LREE and HREE contents of the ore sample are presented in Appendix K. The leaching yielded high individual elemental recovery values in the range 90-92% of the LREE (Figure 4.6). The HREE individual elemental recovery values ranged from 56.5-87.3% (Figure 4.7). In the leaching, 20% HCl was used in the first and second stages, with a duration of 2 hrs in each stage at 90°C. REE contents in the residue of the direct leaching process are shown in Appendix K.

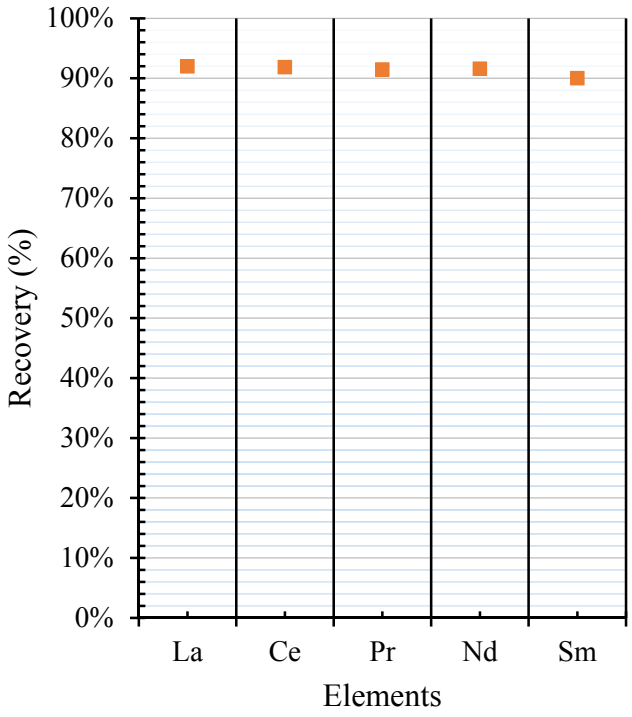

Figure 4.6 Recoveries of LREE as a result of direct leaching

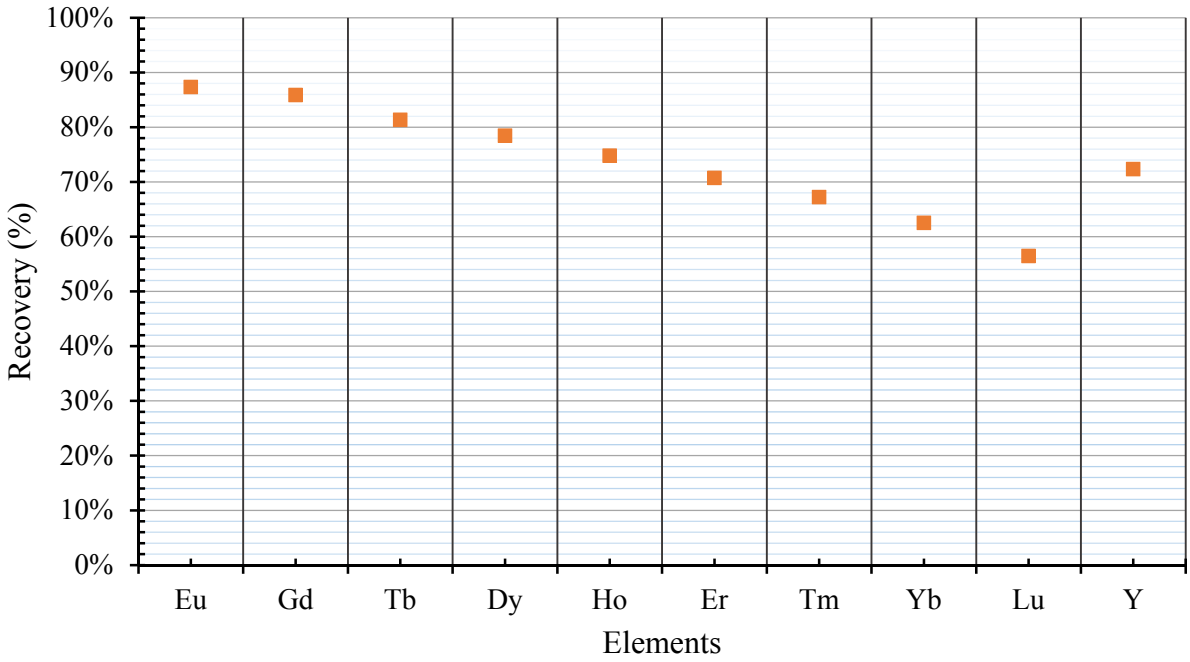

Figure 4.7 Recoveries of HREE as a result of direct leaching

4.4.2 Leach after flotation

For the leach after flotation process, leaching was conducted on concentrate 1 of flotation. REE contents of the feed are shown in Appendix L. Individual elemental recovery values of the LREE and HREE were 94.7-96.5% and 61.1-90.5%, respectively (Figure 4.8 and Figure 4.9). The concentrate was leached using 20% HCl in both the first and second stages, with a duration of 2 hrs in each stage at 90°C. REE contents in the residue of the leach after flotation process are shown in Appendix L. It is assumed that the REE contents in concentrate 1 are similar to that in the combined concentrate of flotation. Flotation/Leach process net recoveries of LREE and HREE by individual elemental values range from 42.2-48.5% and 17.1-41.4%, respectively (Figure 4.10 and Figure 4.11).

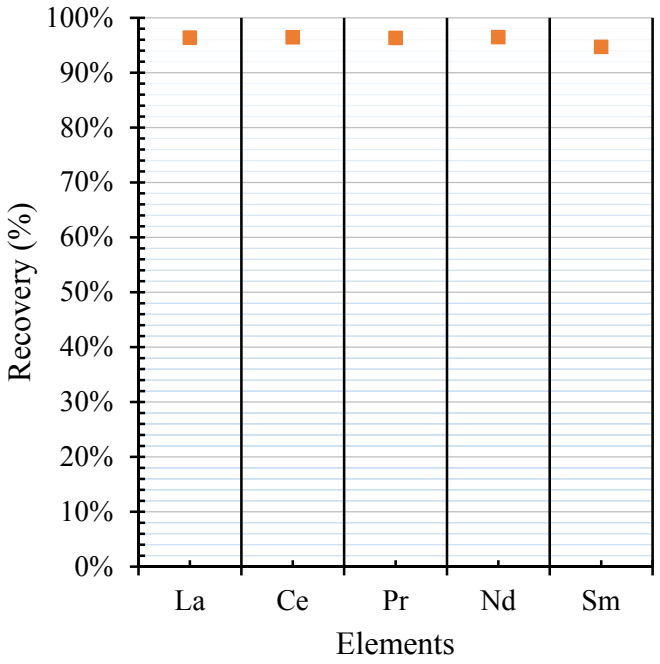

Figure 4.8 Recoveries of LREE as a result of leach after flotation

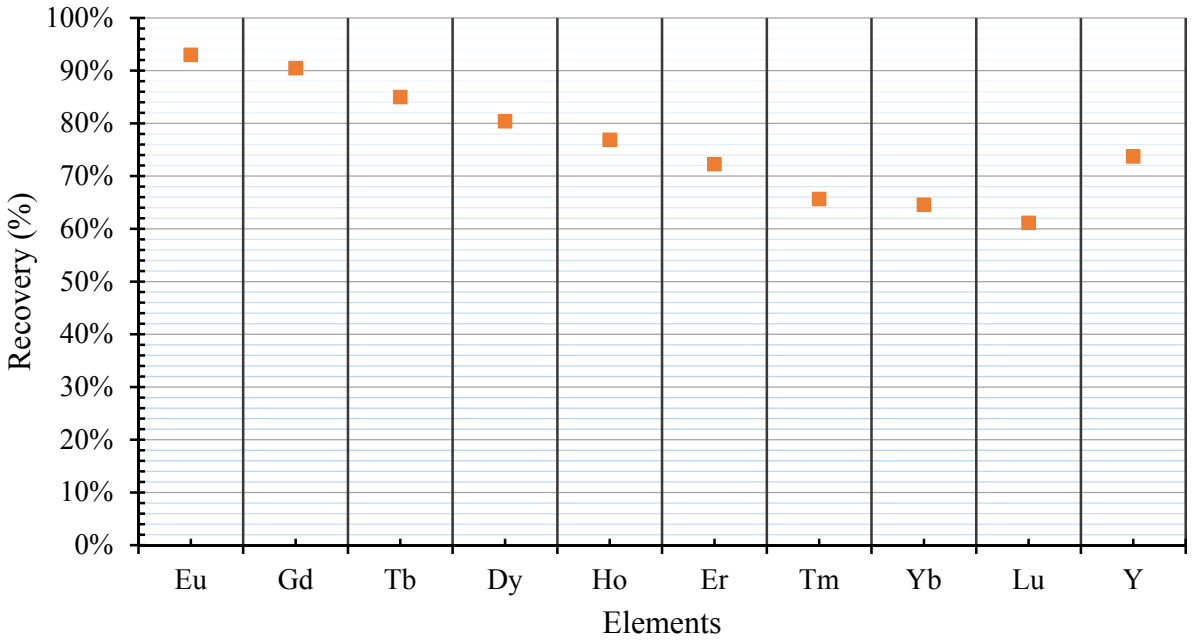

Figure 4.9 Recoveries of HREE as a result of leach after flotation

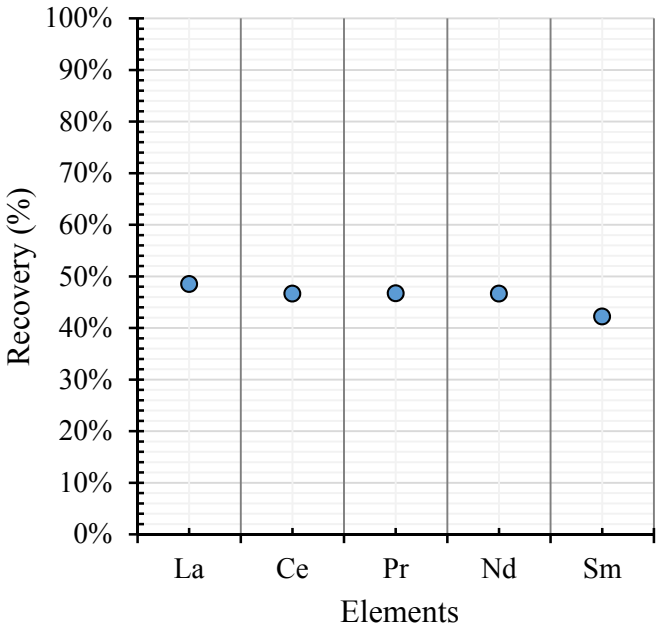

Figure 4.10 Flotation/Leach process net recoveries of LREE

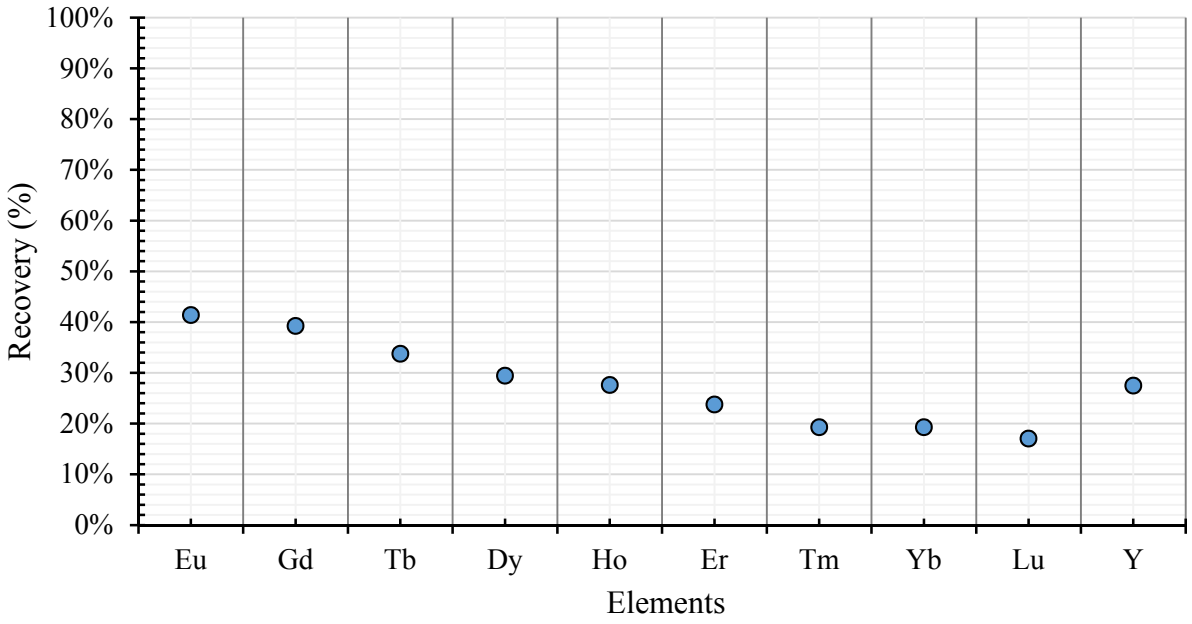

Figure 4.11 Flotation/Leach process net recoveries of HREE

4.5 Electron microprobe analysis

4.5.1 Mineralogy of the ore

After the acquiring process was done, elemental concentration data were provided. The datasheet includes concentration of elements such as Na, Mg Al, Si, P, S, K, Ca, Ti, V, Mn, Fe, Co, Zn, Y, Zr, Nb, Cd, Ba, La, Ce, Pr, Nd, Pm, Sm, Eu, Gd, Tb, Dy, Ho, Er, Tm, Yb, Lu, Hf , Ta, Au, Tl, Pb, Th and U. The REE found in the feed are Y, La, Ce, Pr, Nd, Pm, Sm, Eu, Gd, Tb, Dy, Ho, Er, Tm, Yb and Lu. (Appendix A)

Batch processing of all elemental concentration datasheets was done. As a result of the batch processing, tables of compound concentration data were created. All the compounds were in oxide type, for example, Na_2O, MgO, SiO_2, Fe_2O_3, La_2O_3, Dy_2O_3, ThO_2 and UO_2. Oxygen was determined stoichiometrically (Appendix C).

The compound concentration tables were integrated into one large table for greater ease of accounting and analysis. Also, area data (surface area of the particle) was added to the last table.

As a result of the qualitative analysis, 1600 mineral particles of the compounds were found using the conditions specified in Chapter 3. These mineral particles were found as particles with heavy elements because they were filtered by high atomic number of elements (Chapter 3). 95% of the particle areas of all these mineral particles were smaller than 98 μm^2. This means that the particle size of 95% of the mineral particles was smaller than about 10 μm. Therefore, the particle size of REE mineral particles can be very small.

Since samples were coated with carbon during sample preparation, carbon element was inactivated for analysis. Unfortunately, because of this, carbonate compounds of mineral particles could not be detected.

When La_2O_3 occurs in mineral particles, its maximum concentration can be greater than 10%. This pattern is similar for Ce_2O_3, Nd_2O_3, Dy_2O_3, Y_2O_3 and ZrO_2. Therefore, these are probably the main value compounds that build mineral particles. Note that the proportion of these oxides in the ore is also high, where La, Ce, Nd, Dy and Y contribute 18403 ppm out of 22088 ppm of REE in the

feed ore. Given their contributions, correlations were developed among these six rare earth oxides and other oxide compounds for the mineral particle contents.

Within 1600 mineral particles found, 381 particles had concentrations of La_2O_3 greater than 2%. Maximum La_2O_3 concentration in the mineral particles is 25%. Almost all particles that contained La_2O_3 contained SiO_2. CaO, Ce_2O_3 and Nd_2O_3, and also co-existed with La_2O_3, though their percentage co-existence was lower.

There are 693 particles of minerals with Ce_2O_3 greater than 2%. Maximum Ce_2O_3 concentration is 52.3%. Similar to La_2O_3, Ce_2O_3 co-exists with SiO_2, CaO, Fe_2O_3 and Nd_2O_3 (68-91%), with percentage co-existence being the highest for SiO_2.

For the Nd, there are 654 particles of minerals with Nd_2O_3 greater than 2%. Maximum Nd_2O_3 concentration is 25.7%. In the minerals, Nd_2O_3 co-exists with SiO_2, CaO and Ce_2O_3 (83-96%).

There are 198 particles of minerals with Dy_2O_3 greater than 2% in all the mineral particlesfound. Maximum Dy_2O_3 concentration is 11.9%. In the minerals, most of Dy_2O_3 co-existed with SiO_2 and Y_2O_3 (72-89%).

529 particles of minerals with Y_2O_3 greater than 2% are known in all the mineral particlesfound. Maximum Y_2O_3 concentration is 40.7%. In the minerals, most of Y_2O_3 are co-existed with SiO_2.

324 mineral particles with ZrO_2 greater than 2% are known in all the mineral particlesfound. Maximum ZrO_2 concentration is 53.8%. In the minerals, most of ZrO_2 correlated with SiO_2.

Most of the REE minerals in the ore were silicate minerals. This is not surprising since the ore predominantly contains silicate REE minerals, as determined by UCore and reported here and elsewhere. However, an implication for this is in the use of metasilicate depressant. Too much metasilicate will depress the silicate REM.

P_2O_5 presented co-existence with only Dy_2O_3 and Y_2O_3 of the REE main compounds. However, the co-existence was very low.

Figure 4.12 shows an electron image from microprobe analysis. This image is one of 35 images taken for analysis. Mineral particles with heavy elements are shown as bright shapes. In the selected conditions, 5 particles were recognized as REE minerals. Four mineral particles are highlighted. Two of them, #1 and #2, are comparatively fine particles. The areas of #1 and #2 are 5.4 μm^2 and 2.4 μm^2, respectively (Table 4.2). Particle #1 and #2 have obviously not been liberated completely, as they are inside larger particles.

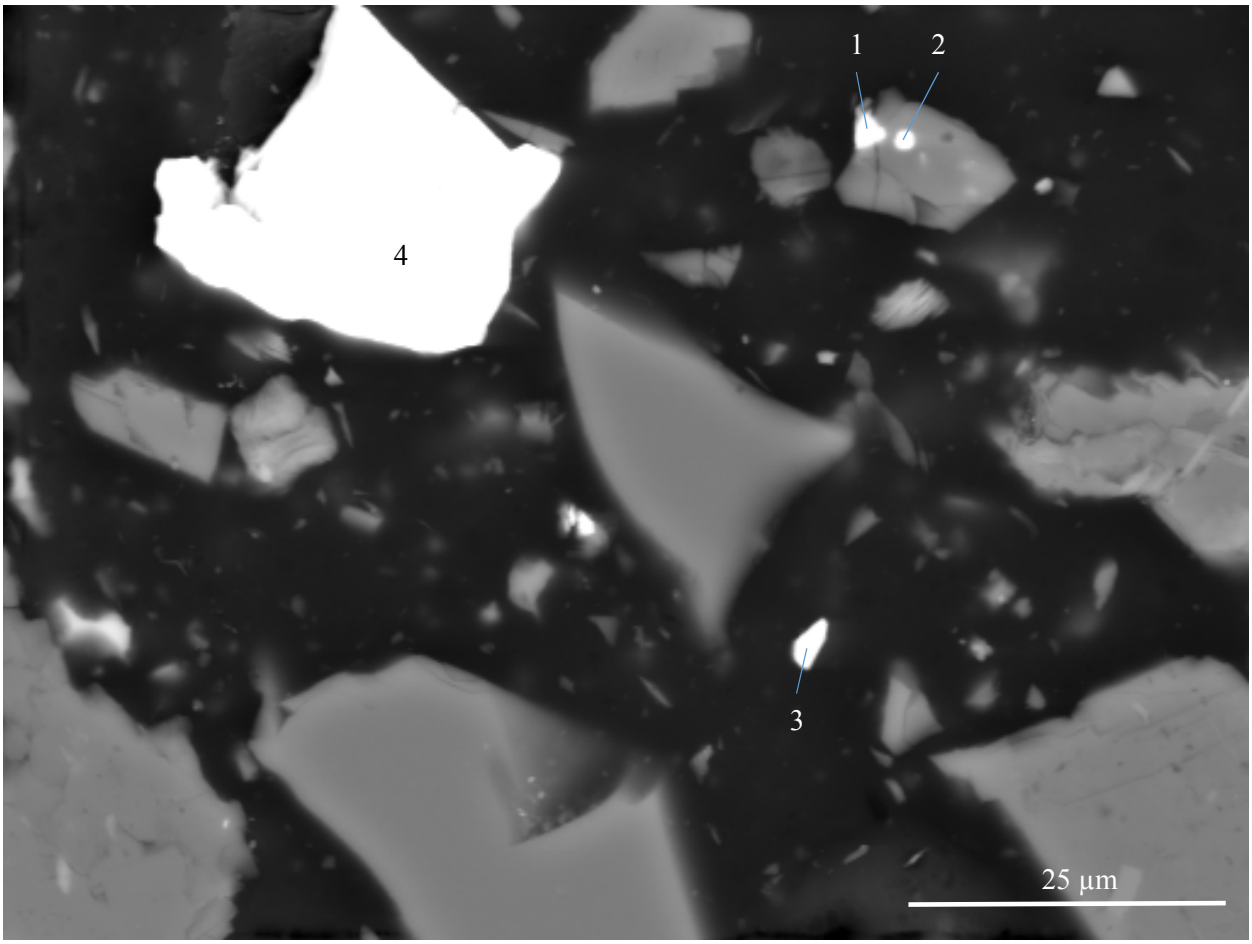

Figure 4.12 Electron image of the feed ore with magnification of 1000X and size of 100x100 microns

Particle #1 is a particle of a REE mineral which has 6.3% Y_2O_3, 18.8% ZrO_2, 6.2% Nb_2O_5, 50.5% SiO_2 and 3.5% P_2O_5 (Table 4.2). Particle #2 is a particle of a REE mineral which has 5.4% La_2O_3, 11.8% Ce_2O_3, 2.8% Nd_2O_3 and 71.3% SiO_2 (Table 4.2). Particle #3 is a fine particle that is completely liberated. Particle #4 is a comparatively big particle that is completely liberated. The areas of particles #3 and #4 are 7.3 μm^2 and 689.2 μm^2, respectively (Table 4.2). Particle #3 is a particle of a REE mineral which has 3.3% La_2O_3, 14% Ce_2O_3, 8.8% Nd_2O_3, 3.2% Sm_2O_3, 3.4% Gd_2O_3, 2.3% Tb_2O_3, 3% Y_2O_3, 2.2% Fe_2O_3, 8% CaO, 15.6% SiO_2, 3.3% MgO and 5% Na_2O (Table 4.2). Particle #4 is a mineral with compounds of 29.9% As_2O_3, 24.5% Fe_2O_3, 22.9% SO_3 and 4.3% MgO (Table 4.2). Particle #4 is not an REE mineral particle. Also, its area is comparatively big. From the area information, it is known that its particle size is about 26 μm. This demonstrates that high atomic number based filtration is not perfect, as it can also highlight non-REE minerals.

Table 4.2 Identification of 4 highlighted mineral particles of feed ore

Particle #	Area (μm^2)	Content of the mineral particle by composition percentage
1	5.4	6.3% Y_2O_3, 18.8% ZrO_2, 6.2% Nb_2O_5, 50.5% SiO_2, 3.5% P_2O_5
2	2.4	5.4% La_2O_3, 11.8% Ce_2O_3, 2.8% Nd_2O_3, 71.3% SiO_2
3	7.3	3.3% La_2O_3, 14% Ce_2O_3, 8.8% Nd_2O_3, 3.2% Sm_2O_3, 3.4% Gd_2O_3, 2.3% Tb_2O_3, 3% Y_2O_3, 2.2% Fe_2O_3, 8% CaO, 15.6% SiO_2, 3.3% MgO, 5% Na_2O
4	689.2	29.9% As_2O_3, 24.5% Fe_2O_3, 22.9% SO_3, 4.3% MgO

4.5.2 Mineralogy of flotation concentrate

After the acquiring process was completed, elemental concentration data were provided. Each datasheet includes concentration of elements such as Na, Mg Al, Si, P, S, K, Ca, Ti, V, Mn, Fe, Co, Zn, Y, Zr, Nb, Cd, Ba, La, Ce, Pr, Nd, Pm, Sm, Eu, Gd, Tb, Dy, Ho, Er, Tm, Yb, Lu, Hf , Ta, Au, Tl, Pb, Th and U. The REE found in the concentrate are Y, La, Ce, Pr, Nd, Pm, Sm, Eu, Gd, Tb, Dy, Ho, Er, Tm, Yb and Lu. (Appendix B)

Batch processing was done for datasheets containing elemental concentration. Tables of compound concentration data were created from batch processing. All the compounds were oxides such as Na_2O, MgO, SiO_2, Fe_2O_3, La_2O_3, Dy_2O_3, ThO_2 and UO_2. Oxygen was determined stoichiometrically. (Appendix D)

The compound concentration tables were integrated into one large table for greater ease of accounting and analysis. Also, area data was added to the last table.

Qualitative analysis identified 1129 mineral particles using conditions specified in Chapter 3. These mineral particles were particles with heavy elements because they were filtered by high atomic number of elements (Chapter 3). 95% of the particle areas of all these mineral particles were smaller than 120 μm^2. This means that 95% of the mineral particles were smaller than about 11 μm. Therefore, the particle size of REE mineral particles can be very small.

When La_2O_3 occurs in mineral particles, its maximum concentrations can be greater than 10%. This pattern is similar for Ce_2O_3, Nd_2O_3, Dy_2O_3, Nb_2O_5, Y_2O_3 and ZrO_2. Therefore, these are probably the main value compounds which build mineral particles of the concentrate of flotation. Therefore, as with the concentrate, each of these elements was investigated for co-existence with all other minerals.

Four hundred fifty five particles had concentrations of La_2O_3 greater than 2%. Maximum La_2O_3 concentration is 30.2%. In the compounds, most La_2O_3 co-exists with SiO_2. Furthermore, La_2O_3 has good co-existence with CaO, Fe_2O_3, Y_2O_3, Ce_2O_3, Pr_2O_3, Nd_2O_3, Pm_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3 and Tb_2O_3.

Also, there are 750 particles of compounds with Ce_2O_3 greater than 2%. Maximum Ce_2O_3 concentration is 54.6%. In the compounds, most of Ce_2O_3 were co-existed with SiO_2. Besides, La_2O_3 had good co-existence with CaO, Fe_2O_3, Y_2O_3, La_2O_3, Pr_2O_3, Nd_2O_3, Pm_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3 and Tb_2O_3.

For Nd, there are 714 particles of compounds with Nd_2O_3 greater than 2%. Maximum Nd_2O_3 concentration is 22.6%. In the compounds, most Nd_2O_3 co-existed with SiO_2. In addition, Nd_2O_3 has good co-existence with CaO, Fe_2O_3, Y_2O_3, La_2O_3, Ce_2O_3, Pr_2O_3, Pm_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3 and Tb_2O_3.

There are 97 particles of compounds with Dy_2O_3 greater than 2%. Maximum Dy_2O_3 concentration is 10.5%. In the compounds, most Dy_2O_3 co-existed with SiO_2. Dy_2O_3 has good co-existence with CaO, Fe_2O_3, Y_2O_3, Ce_2O_3, Pr_2O_3, Nd_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3, Tb_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3 and Lu_2O_3.

There are 101 particles of compounds with Nb_2O_5 greater than 2%. Maximum Nb_2O_5 concentration is 55.6%. In the compounds, most Nb_2O_5 co-existed with SiO_2. Nb_2O_5 has good co-existence with CaO, Fe_2O_3, Y_2O_3, Ce_2O_3, Pr_2O_3, Nd_2O_3, Pm_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3, Tb_2O_3, Dy_2O_3 and Tm_2O_3.

518 particles of compounds with Y_2O_3 greater than 2% are known. Maximum Y_2O_3 concentration is 92.1%. In the compounds, most Y_2O_3 co-existed with SiO_2. Y_2O_3 has good co-existence with CaO, Fe_2O_3, La_2O_3, Ce_2O_3, Pr_2O_3, Nd_2O_3, Pm_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3, Tb_2O_3 and Dy_2O_3.
45 particles of compounds with ZrO_2 greater than 2% are known. Maximum ZrO_2 concentration is 57.8%. In the compounds, all of the ZrO_2 co-existed with SiO_2. Most of ZrO_2 are co-existed with P_2O_5. ZrO_2 has good co-existence with Fe_2O_3, Y_2O_3, Ce_2O_3, Nd_2O_3, Tb_2O_3 and Er_2O_3.

There are 18 particles of compounds with ThO_2 greater than 2%. Maximum ThO_2 concentration in the compounds is 43.7%. In the compounds, all ThO_2 co-existed with SiO_2. Also, ThO_2 has good co-existence with CaO, Fe_2O_3, Y_2O_3, La_2O_3, Ce_2O_3, Pr_2O_3, Nd_2O_3, Pm_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3 and Tb_2O_3.

There are 115 particles of compounds with UO_2 greater than 2%. Maximum UO_2 concentration is 11.2%. In the compounds, most UO_2 co-existed with SiO_2. UO_2 has good co-existence with CaO, Fe_2O_3, Y_2O_3, Ce_2O_3, Nd_2O_3, Pm_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3 and Tb_2O_3.

The results of the qualitative analysis show that REE minerals containing La, Ce, Nd, Dy, Nb, Y, Zr, Th and U were separated into the concentrate by flotation process. There were probably main minerals of La, Ce, Nb, Y, Zr and Th in the concentrate. Most of the REE minerals separated into the concentrate were silicate minerals. Note that co-existence of these minerals with phosphate minerals was explored, but nothing significant was found.

Thirty six images were taken for electron microprobe analysis of flotation concentrate particles. Figure 4.13 shows one of those images. Mineral particles with heavy elements are presented as bright shapes. In the selected conditions, 35 particles were recognized as REE minerals. Six mineral particles are highlighted on Figure 4.13. Two of them, #1 and #2, are comparatively big particles. The areas of #1 and #2 are 362.8 μm^2 and 285.6 μm^2, respectively (Table 4.3). Particle #1 is a particle of a REE mineral which has 14% La_2O_3, 42.8% Ce_2O_3, 12.9% Nd_2O_3, 2.6% Sm_2O_3, 1.2% Eu_2O_3, 2.7% Gd_2O_3, 2.9% Y_2O_3, 13.4% CaO and 4.4% SiO_2 (Table 4.3). Particle #2 is a particle of a REE mineral which has 1.4% La_2O_3, 5.9% Ce_2O_3, 3.2% Nd_2O_3, 2.6% Sm_2O_3, 2.2% Eu_2O_3, 2.2% Gd_2O_3, 6.7% Dy_2O_3, 2.0% Ho_2O_3, 23.1% Y_2O_3, 2.8% Fe_2O_3 and 38.8% SiO_2 (Table 4.3). Particles #3 and #4 are very fine particles that are completely liberated. The areas of #3 and #4 are 2.7 μm^2 and 1.5 μm^2, respectively (Table 4.3). Particle #3 is a particle of a REE mineral which has 2.5% La_2O_3, 13.4% Ce_2O_3, 7.1% Nd_2O_3, 1.5% Sm_2O_3, 1.2% Gd_2O_3, 1.6% Tb_2O_3, 2.7% Y_2O_3, 4.9% Fe_2O_3, 8.6% CaO, 1.3% K_2O, 34.7% SiO_2, 6.1% MgO and 8.7% Na_2O (Table 4.3).

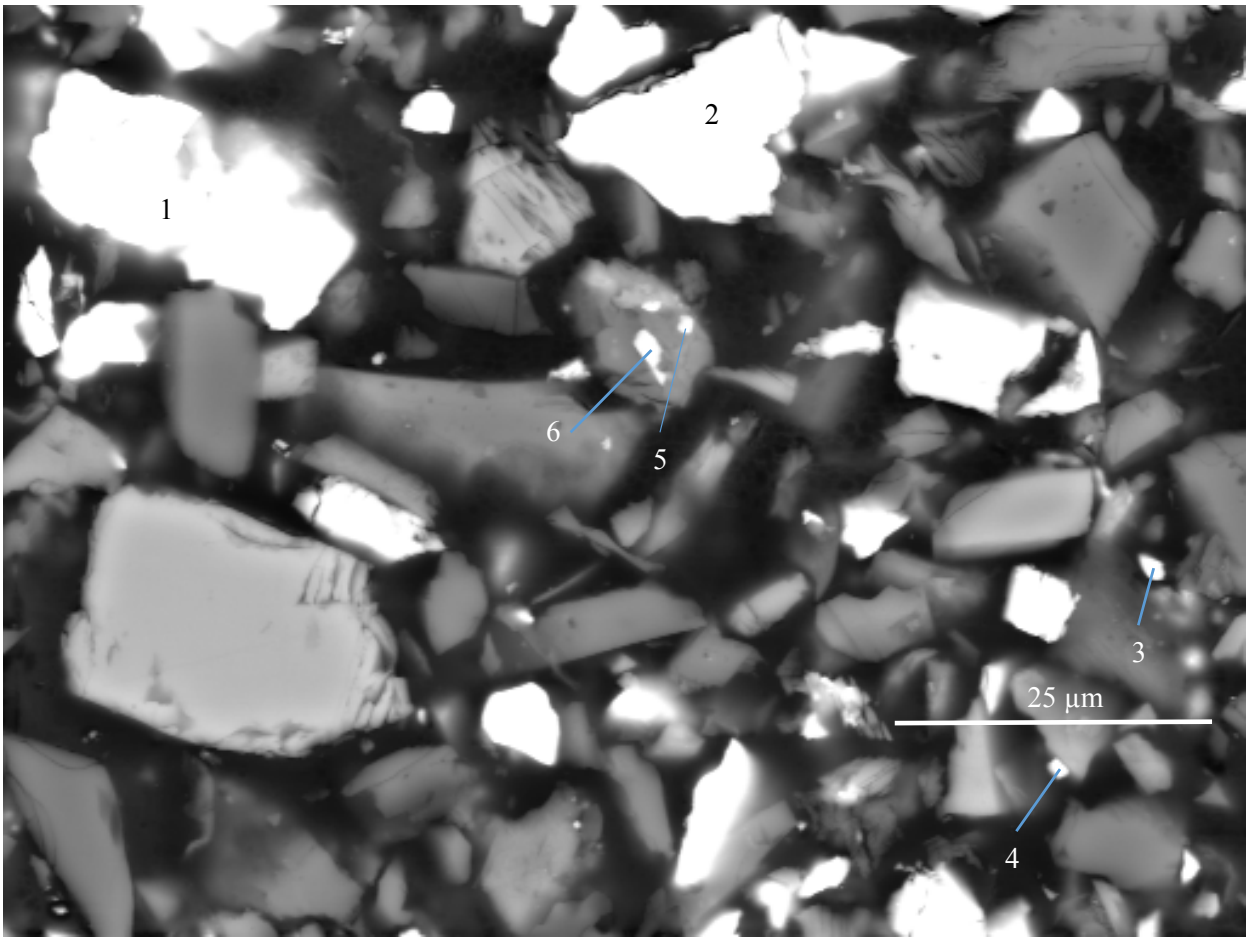

Figure 4.13 Electron image No. 1 of the flotation concentrate with magnification of 1000X and size of 100x100 microns

Table 4.3 Identification of 6 of highlighted mineral particles in electron image No. 1

Particle # on the figure	Area (μm^2)	Content by composition percentage
1	362.8	14% La_2O_3, 42.8% Ce_2O_3, 12.9% Nd_2O_3, 2.6% Sm_2O_3, 1.2% Eu_2O_3, 2.7% Gd_2O_3, 2.9% Y_2O_3, 13.4% CaO, 4.4% SiO_2
2	285.6	1.4% La_2O_3, 5.9% Ce_2O_3, 3.2% Nd_2O_3, 2.6% Sm_2O_3, 2.2% Eu_2O_3, 2.2% Gd_2O_3, 6.7% Dy_2O_3, 2.0% Ho_2O_3, 23.1% Y_2O_3, 2.8% Fe_2O_3, 38.8% SiO_2
3	2.7	2.5% La_2O_3, 13.4% Ce_2O_3, 7.1% Nd_2O_3, 1.5% Sm_2O_3, 1.2% Gd_2O_3, 1.6% Tb_2O_3, 2.7% Y_2O_3, 4.9% Fe_2O_3, 8.6% CaO, 1.3% K_2O, 34.7% SiO_2, 6.1% MgO, 8.7% Na_2O
4	1.5	6.4% Ce_2O_3, 4.0% Nd_2O_3, 1.6% Sm_2O_3, 1.6% Eu_2O_3, 1.1% Y_2O_3, 1.6% UO_2, 16.3% Fe_2O_3, 3.9% CaO, 5.7% K_2O, 1.0% SO_3, 36.4% SiO_2, 15.1% MgO,
5	1.3	37.9% Fe_2O_3, 6.3% CaO, 2.0% K_2O, 40.3% SiO_2, 5.5% MgO, 4.7% Na_2O
6	4.6	40.4% Fe_2O_3, 3.4% CaO, 52.8% SiO_2

In the Figure 4.13, the particles #5 and #6 look like REE mineral grains which are not liberated. However, it was known that these minerals were not REE's. The #5 is a mineral with compounds of 37.9% Fe_2O_3, 6.3% CaO, 2.0% K_2O, 40.3% SiO_2, 5.5% MgO and 4.7% Na_2O (Table 4.3). The #6 is a mineral which has 40.4% Fe_2O_3, 3.4% CaO and 52.8% SiO_2 (Table 4.3). Perhaps, the threshold setting of the filtration by high atomic number of elements probably was not sufficient.

In Figure 4.14, an electron image of mineral particles of flotation concentrate is shown. Mineral particles with heavy elements are presented as bright shapes. In the selected conditions, 22 particles were recognized as REE minerals. All of them have areas lower than 53 μm^2 or sizes lower than about 7 μm. Six mineral particles are highlighted on Figure 4.14. Particle #1 is a particle of a REE mineral which has 3.0% La_2O_3, 15.6% Ce_2O_3, 7.8% Nd_2O_3, 2.9% Sm_2O_3, 2.1% Gd_2O_3, 1.7% Tb_2O_3, 4.5% Y_2O_3, 2.5% Fe_2O_3, 6.9% CaO and 49.8% SiO_2 (Table 4.4). Particle #2 is a particle of a REE mineral which has 4.5% La_2O_3, 16.9% Ce_2O_3, 1.8% Pr_2O_3, 7.5% Nd_2O_3, 1.6% Sm_2O_3, 2.2% Gd_2O_3, 1.5% Tb_2O_3, 6.9% Y_2O_3, 5.9% Fe_2O_3, 9.6% MgO, 10.0% CaO and 27.6% SiO_2 (Table 4.4). Particle #4 is a particle of a REE mineral which has 7.4% La_2O_3, 31.8% Ce_2O_3, 12.0% Nd_2O_3, 4.3% Sm_2O_3, 1.5% Eu2O3, 1.8% Gd_2O_3, 2.8% Y_2O_3, 3.6% Fe_2O_3, 6.4% CaO and 25.0% SiO_2. Particle #6 is a particle of a REE mineral which has 1.5% La_2O_3, 7.1% Ce_2O_3, 1.1% Pr2O3, 3.8% Nd_2O_3, 1.0% Sm_2O_3, 2.4% Y_2O_3, 3.3% CaO and 77.1% SiO_2. The content of particle #5 is not identified in a specified condition (Chapter 3) because its area is lower than 1 μm^2 (Table 4.4). Particle #3 is not a particle of REE mineral (Table 4.4).

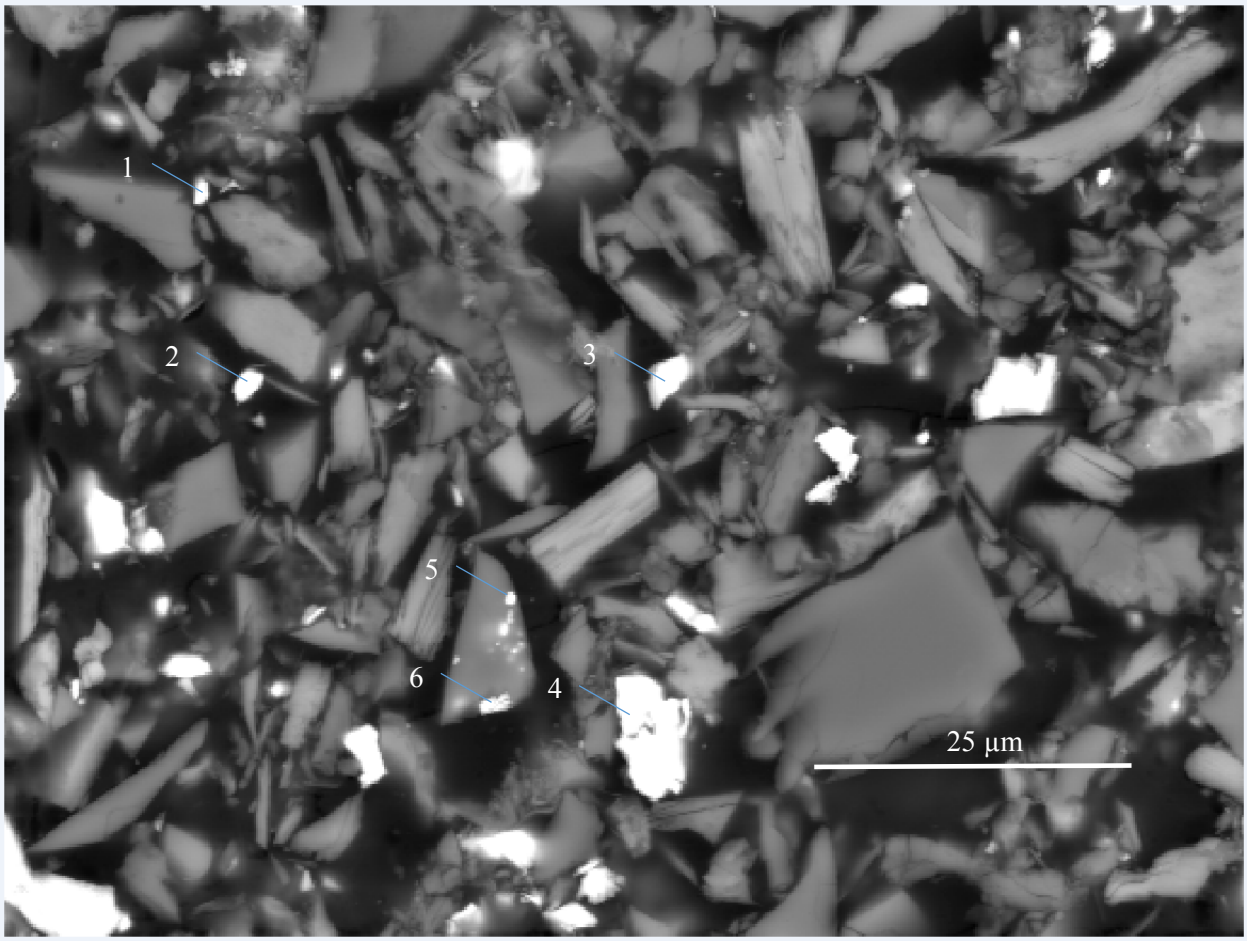

Figure 4.14 Electron image No. 2 of the flotation concentrate with magnification of 1000X and size of 100x100 microns

Table 4.4 Identification of 6 of highlighted mineral particles in the electron image No. 2

Particle # on the figure	Area (μm^2)	Content by composition percentage
1	1.80	3.0% La_2O_3, 15.6% Ce_2O_3, 7.8% Nd_2O_3, 2.9% Sm_2O_3, 2.1% Gd_2O_3, 1.7% Tb_2O_3, 4.5% Y_2O_3, 2.5% Fe_2O_3, 6.9% CaO and 49.8% SiO_2
2	4.29	4.5% La_2O_3, 16.9% Ce_2O_3, 1.8% Pr_2O_3, 7.5% Nd_2O_3, 1.6% Sm_2O_3, 2.2% Gd_2O_3, 1.5% Tb_2O_3, 6.9% Y_2O_3, 5.9% Fe_2O_3, 9.6% MgO, 10.0% CaO and 27.6% SiO_2
3	8.37	45% ZrO_2, 1.6% Fe_2O_3, 11.4% P_2O_5, 36.3% SiO_2 and 2.5% MgO
4	52.58	7.4% La_2O_3, 31.8% Ce_2O_3, 12.0% Nd_2O_3, 4.3% Sm_2O_3, 1.5% Eu2O3, 1.8% Gd_2O_3, 2.8% Y_2O_3, 3.6% Fe_2O_3, 6.4% CaO and 25.0% SiO_2
5	<1	N/A
6	1.59	1.5% La_2O_3, 7.1% Ce_2O_3, 1.1% Pr2O3, 3.8% Nd_2O_3, 1.0% Sm_2O_3, 2.4% Y_2O_3, 3.3% CaO and 77.1% SiO_2

Chapter 5 Conclusions and Recommendation

5.1 Conclusions

The REE ore samples from the Bokan Mountain – Dotson Ridge deposit were subjected to gravity concentration, magnetic separation, flotation, and leaching techniques to recover the REE. Additionally, electron microprobe analysis was done to understand the mineral processing results. The major findings of the research work are provided below:

The results of the electron microprobe analysis show that La_2O_3, Ce_2O_3, Nd_2O_3, Dy_2O_3, Y_2O_3 and ZrO_2 are probably the main value compounds building the mineral particles in the feed composite ore. Most of the REE minerals in the ore were silicate minerals. 95% of the REE mineral particles in the ore are smaller than about 10 μm. Most Dy_2O_3 co-existed with SiO_2 and Y_2O_3 in its minerals.

For the gravity concentration tests, the LREE and HREE individual elemental recovery values were in the ranges of 49.6-52.8% and 46.3-48.8%, respectively, at 25% mass yield. In order to separate more of the REE, the gravity separation tailings stream was subjected to wet high intensity magnetic separation. For the magnetic separation, the HREE individual elemental recovery values ranged from 56.3-63.1% at 37% mass yield. For the LREE, they were separated from the tailings of the gravity separation with individual elemental recovery values ranging from 57.9-59.1%. For the combined gravity and magnetic separation process, net individual elemental recovery values of the LREE and the HREE were in the ranges of 79.6-80.5% and 76.5-80.9%, respectively. The combined mass yield of the Gravity/Magnetic separation process was 53%.

Direct leaching tests were conducted on the composite ground ore feed. The leaching yielded high individual elemental recovery values in the range of 90-92% for the LREE. The HREE individual elemental recovery values ranged from 56.5-87.3%. In the leaching, 20% HCl was used in the first and second stages, with a duration of 2 hrs in each stage at 90°C.

The composite ground ore sample was tested in conventional flotation using a 2.0 L capacity Denver cell. In flotation, 0.05 kg/tonne of Cytec Aero 6493 collector, 0.05 kg/tonne of Cytec Aero

Froth 88 and 0.1 kg/tonne of sodium metasilicate depressant were used at a pH around 9. Results showed individual elemental recovery values in the range of 44.6-50.4% for the LREE at a mass yield of 23%. The HREE individual elemental recovery values ranged from 27.9-44.5%. For the leach after flotation process, leaching was conducted on concentrate 1 of flotation. Individual elemental recovery values of the LREE and HREE were 94.7-96.5% and 61.1-90.5%, respectively. The concentrate was leached using 20% HCl in both the first and second stages with a duration of 2 hrs in each stage at 90°C. If it is assumed that the REE contents in concentrate 1 are similar to those in the combined concentrate of flotation, the combined flotation and leach process net recoveries of LREE and HREE by individual elemental values ranged from 42.2-48.5% and 17.1-41.4%, respectively. For the flotation, and combined flotation and leach process, recoveries in this thesis were significantly lower than what was achieved previously.

As a result of the electron microprobe analysis, it was shown that REE minerals containing value elements La, Ce, Nd, Dy, Nb, Y and Zr were separated into the concentrate by the flotation process. La_2O_3, Ce_2O_3, Nb_2O_5, Y_2O_3 and ZrO_2 were probably the main value compounds building mineral particles of the concentrate of flotation. 95% of the REE mineral particles in the concentrate might be smaller than about 11 μm. Most of the REE minerals separated into the concentrate were silicate minerals.

5.2 Recommendation for future work

HREE individual elemental recovery values were lower than those of LREE. Perhaps HREE mineral particles were non-liberated. Therefore, regrinding residue of the leaching and new stage of leaching are recommended.

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Appendix A Elemental concentrations (%) of the first 48 mineral particles of 1600 particles of the feed ore as result of e-probe analysis

Particles #	O	F	Na	Mg	Al	Si	P	S	K	Ca	Ti	V	Mn	Fe	Co	Cu	Zn	As	Y	Zr	Nb	Mo	Cd	Sn	Ba	La
A	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
Feed_Particle 000001	32.1	0.0	1.2	1.9	0.0	15.0	0.0	0.0	0.0	7.4	0.0		3.5	8.4		0.0	0.1	0.0	0.8	1.1	0.9	0.8		0.0		1.8
Feed_Particle 000002	21.7	0.0	5.8	0.4	0.0	2.3	0.2	0.0	0.2	13.1	0.1		0.2	0.8		0.1	0.0	0.0	3.9	2.1	1.1	2.2		0.0		4.8
Feed_Particle 000003	38.8	0.0	0.0	0.0	0.0	10.4	0.0	0.0	0.1	17.7	28.2	0.2	0.1	2.0		0.0	0.2	0.0	0.2	0.4	0.0	1.2		0.0		0.0
Feed_Particle 000004	38.2	0.0	0.0	3.9	0.0	11.7	0.0	0.0	0.1	14.3	16.8	0.1	0.8	10.1		0.1	0.1	0.0	0.2	0.0	0.4	0.0		0.4		0.0
Feed_Particle 000005	28.8	0.0	2.5	10.4	0.0	8.9	0.0	0.0	0.2	0.3	0.0		0.3	8.2		0.0	0.0	0.1	0.4	0.4	1.0	0.0		0.0		6.0
Feed_Particle 000006	30.9	0.0	1.2	1.1	0.0	13.6	0.0	0.2	1.0	4.1	0.0		0.0	6.9		0.0	0.5	0.0	0.0	2.7	2.2	4.6		1.1		1.4
Feed_Particle 000007	30.9	0.0	0.0	0.0	1.6	3.0	0.0	0.0	0.1	2.1	22.7	0.0	16.6	9.8		0.0	0.0	0.0	0.0	0.0	0.0	0.0		0.1		0.0
Feed_Particle 000008	29.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	1.3	24.9	0.1	24.2	9.2		0.0	0.1	0.0	0.5	0.2	0.0	0.0		0.2		0.5
Feed_Particle 000009	31.7	0.0	1.2	2.8	0.0	13.3	0.0	0.0	0.1	9.4	0.0		1.3	9.8		0.0	0.2	0.0	0.0	1.9	0.0	1.5		0.0		2.5
Feed_Particle 000010	42.4	0.0	2.3	0.0	0.0	23.5	4.7	0.0	0.0	0.0	0.0		0.0	0.9		0.0	0.2	0.0	0.0	20.3	0.1	0.8		0.3		0.1
Feed_Particle 000011	33.0	0.0	2.4	2.0	0.0	15.2	0.0	0.0	0.0	9.5	0.0		0.7	11.3		0.0	0.4	0.1	2.3	0.0	0.6	0.0		0.0		1.6
Feed_Particle 000012	31.9	0.0	1.4	2.1	0.0	14.0	0.1	0.0	0.1	6.4	0.0		2.5	10.6		0.0	0.0	0.2	0.7	0.0	1.0	1.6		0.6		3.0
Feed_Particle 000013	47.8	0.0	0.0	0.0	0.1	39.3	0.0	0.0	0.0	2.0	0.0		0.1	0.2		0.0	0.1	0.0	0.7	0.0	0.5	0.0		0.3		1.3
Feed_Particle 000014	32.7	0.0	7.5	1.0	0.0	7.4	0.0	0.2	0.2	0.4	0.0		0.0	48.9		0.0	0.0	0.0	0.2	0.4	1.8	0.0		0.4		0.0
Feed_Particle 000015	45.7	0.0	1.0	0.0	0.0	30.9	3.5	0.0	0.1	0.0	0.1		0.0	0.1		0.0	0.2	0.0	1.3	11.4	0.6	0.6		0.0		0.3
Feed_Particle 000016	39.6	0.0	1.7	0.0	0.0	23.6	1.5	0.0	0.2	0.5	0.0		0.3	0.8		0.0	0.0	0.0	4.9	13.9	4.3	0.3		0.0		0.3
Feed_Particle 000017	48.0	0.0	0.0	0.0	0.0	33.3	0.0	0.2	0.0	0.5	0.1		0.0	0.2		0.1	0.1	0.0	0.3	0.0	0.0	1.5		0.1		4.6
Feed_Particle 000018	30.3	17.2	0.0	2.6	0.0	0.0	0.0	9.2	0.0	0.0	0.0		0.0	17.2		0.0	0.0	22.6	0.0	0.3	0.0	0.0		0.0		0.0
Feed_Particle 000019	40.9	0.0	3.7	2.6	0.0	22.5	0.0	0.0	0.0	0.0	0.0		0.1	27.2		0.1	0.0	0.1	0.7	0.0	0.0	0.0		0.1		0.1
Feed_Particle 000020	21.4	16.9	3.7	2.0	0.0	7.3	0.0	0.1	0.9	5.7	0.0		0.4	1.6		0.0	0.1	0.3	2.3	0.5	0.7	0.0		0.0		2.3
Feed_Particle 000021	30.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.1	3.3	0.0	0.0	62.3		0.0	0.0	0.0	0.0	0.7	0.0	0.0		0.4		0.0
Feed_Particle 000022	30.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	3.2	0.0	0.2	63.6		0.0	0.1	0.0	0.2	0.0	0.0	0.0		0.0		0.0
Feed_Particle 000023	23.3	0.0	3.1	0.0	0.0	7.3	0.0	0.0	0.3	9.7	0.0		0.1	0.5		0.0	0.6	0.0	3.0	0.2	0.5	0.0		0.0		4.9
Feed_Particle 000024	32.7	0.0	12.1	0.4	0.0	12.6	0.0	0.0	0.2	1.8	3.4		1.0	6.4		0.0	0.2	0.0	6.5	1.3	7.7	1.0		0.0		0.0
Feed_Particle 000025	37.4	0.0	2.4	0.0	0.0	13.9	5.7	0.0	0.2	2.6	0.0		0.4	3.0		0.3	0.0	0.0	1.7	21.3	1.2	2.3		0.0		0.3
Feed_Particle 000026	33.3	0.0	7.8	2.5	0.0	14.3	0.0	0.1	0.5	1.9	0.0		0.7	2.8		0.1	0.3	0.5	6.9	1.0	12.7	0.7		0.0		0.3
Feed_Particle 000027	30.8	0.0	3.7	0.8	0.0	3.8	0.1	0.1	0.1	4.1	0.1		0.0	49.2		0.0	0.0	0.0	0.9	0.4	0.4	0.0		0.0		0.6
Feed_Particle 000028	33.5	0.0	1.3	1.7	0.0	14.5	1.4	0.4	0.2	5.2	0.0		1.2	5.7		0.0	0.2	0.0	3.1	9.1	1.3	0.2		0.0		0.2
Feed_Particle 000029	25.4	0.0	7.2	0.4	0.0	8.4	0.3	0.2	0.3	8.1	0.0		0.2	1.0		0.0	0.5	0.1	1.5	1.1	0.0	0.9		0.0		2.9
Feed_Particle 000030	33.4	0.0	3.5	3.2	0.0	14.7	0.0	0.0	1.1	2.3	0.3		0.2	4.4		0.0	0.0	0.1	5.1	6.5	8.4	0.0		0.0		1.8
Feed_Particle 000031	42.3	0.0	1.8	0.0	0.0	32.9	0.0	0.0	0.0	0.2	0.0		0.0	0.4		0.0	0.1	0.1	1.2	0.3	0.0	0.0		0.1		2.0
Feed_Particle 000032	41.1	0.0	1.8	0.0	0.0	29.0	0.0	0.0	0.0	4.2	0.0		0.9	5.0		0.1	0.0	0.0	0.0	0.6	0.4	0.5		0.0		1.3
Feed_Particle 000033	30.9	0.0	1.7	0.3	0.0	4.6	0.0	0.0	0.2	0.5	0.2		0.2	48.4		0.0	0.1	0.0	2.4	1.3	3.9	0.0		0.0		0.3
Feed_Particle 000034	33.9	0.0	3.0	3.1	0.0	17.4	0.0	0.0	0.0	3.7	0.0		1.9	9.2		0.0	0.0	0.3	0.2	0.0	1.1	0.2		0.0		1.3
Feed_Particle 000035	39.6	0.0	1.6	1.5	0.0	25.2	0.0	0.0	0.0	7.4	0.0		1.1	7.2		0.1	0.1	0.0	0.8	0.0	0.2	0.2		0.0		0.3
Feed_Particle 000036	39.6	0.0	3.5	0.0	0.0	20.0	3.6	0.4	0.1	0.0	0.1		0.1	1.2		0.0	0.3	0.0	3.3	21.6	0.0	0.3		0.1		0.4
Feed_Particle 000037	39.5	0.0	2.5	0.0	0.0	23.9	0.0	0.0	0.0	0.1	1.2		3.7	2.4		0.0	0.0	0.0	2.0	1.9	13.9	0.0		0.0		0.0
Feed_Particle 000038	26.7	0.0	8.3	0.6	0.0	10.3	0.0	0.1	0.1	7.1	0.0		0.6	0.9		0.2	0.1	0.0	1.3	1.1	0.8	0.0		0.0		3.6
Feed_Particle 000039	38.0	0.0	11.9	2.2	0.0	22.8	0.0	0.0	0.0	0.0	0.1		0.1	2.8		0.0	0.1	0.0	11.9	0.8	0.0	0.7		0.2		0.0
Feed_Particle 000040	23.4	0.0	4.5	0.0	0.0	7.2	0.0	0.0	0.1	8.5	0.0		0.5	0.8		0.0	0.4	0.0	2.8	1.2	0.0	0.0		0.0		2.2
Feed_Particle 000041	32.1	0.0	3.1	0.0	0.0	15.0	0.0	0.0	0.1	11.2	0.1		0.4	10.0		0.0	0.0	0.0	0.0	2.1	0.0	0.0		0.0		1.2
Feed_Particle 000042	40.8	0.0	1.8	0.0	0.0	17.6	6.0	0.4	0.0	1.0	0.1		0.0	0.6		0.0	0.0	0.0	1.3	24.4	1.4	2.5		0.0		0.0
Feed_Particle 000043	40.6	0.0	3.9	0.0	0.0	26.3	0.6	0.3	0.0	1.0	3.5		1.0	3.5		0.0	0.0	0.0	0.0	1.7	0.5	0.0		0.5		0.5
Feed_Particle 000044	25.6	0.0	0.0	2.5	0.0	5.5	0.0	0.1	3.5	0.9	0.2		0.2	5.3		0.0	0.0	0.0	0.0	2.2	13.8	1.6		0.0		0.4
Feed_Particle 000045	37.5	0.0	2.0	1.2	0.0	11.3	7.2	0.4	0.6	0.7	0.0		0.0	2.0		0.0	0.2	0.0	3.0	21.4	2.5	2.7		0.3		0.0
Feed_Particle 000046	34.8	0.0	0.0	0.0	3.5	8.3	0.0	0.0	0.3	12.3	8.8	0.0	0.1	6.9		0.0	0.0	0.0	0.0	2.0	14.9	0.1		0.0		0.0
Feed_Particle 000047	25.0	0.0	2.5	0.0	0.0	4.5	0.0	0.0	0.2	1.2	0.8		0.0	1.6		0.0	0.2	0.1	19.7	0.1	18.0	1.1		0.0		0.4
Feed_Particle 000048	37.2	23.4	0.0	0.0	0.3	0.0	0.0	19.3	0.1	0.1	0.0		0.0	17.0		0.0	0.0	0.2	0.0	0.6	0.4	0.0		0.0		0.0

Appendix A Elemental concentrations (%) of the first 48 mineral particles of the feed ore as result of e-probe analysis (continued)

Particles #	Ce	Pr	Nd	Pm	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Ta	Au	Pb	Th	U
A	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45
Feed_Particle 000001	8.3	1.0	4.6	0.9	2.2	2.0	2.4	2.7	0.3	0.4	0.0	0.2	0.0	0.0	0.0		0.0	0.0	0.0
Feed_Particle 000002	16.8	2.4	9.4	0.7	2.0	0.5	3.2	1.7	0.9	0.0	0.0	0.2	0.0	0.5	0.0		0.0	0.0	2.8
Feed_Particle 000003	0.0	0.3	0.1	0.0	0.2	0.0	0.0	0.0	0.0	0.0	0.1	0.4	0.1	0.0	0.0		0.5	0.4	0.3
Feed_Particle 000004	0.0	0.0	0.0	0.0	0.3	0.6	0.2	0.0	0.0	0.0	0.0	0.2	0.2	0.0	0.0		0.5	1.0	0.0
Feed_Particle 000005	16.0	2.2	7.0	1.1	1.9	0.3	1.3	1.7	0.0	0.0	0.0	0.1	0.0	0.0	0.0		0.1	0.0	0.7
Feed_Particle 000006	4.9	0.0	1.3	1.2	0.0	0.0	0.6	2.4	0.0	0.0	1.9	2.1	1.6	1.5	0.0		0.0	3.1	7.6
Feed_Particle 000007	0.5	1.0	0.4	0.0	0.0	7.8	2.5	3.2	0.0	0.0	0.0	0.0	0.0	0.4	0.0		0.0	1.1	0.0
Feed_Particle 000008	0.5	0.9	0.4	0.5	1.1	0.1	0.8	1.7	0.7	0.4	0.0	0.0	0.2	0.0	0.1		0.6	0.8	0.0
Feed_Particle 000009	10.9	1.8	4.8	0.0	0.6	0.6	0.5	1.5	0.0	0.0	0.0	0.4	0.0	0.0	0.0		0.0	0.4	2.6
Feed_Particle 000010	0.2	0.1	0.2	0.0	0.0	0.0	0.0	0.3	0.1	0.2	0.0	0.0	0.0	0.0	0.0		0.3	1.4	1.4
Feed_Particle 000011	9.5	1.2	4.3	0.3	1.4	0.4	0.6	1.6	0.0	0.0	0.0	0.1	0.0	0.0	0.0		0.2	0.0	1.2
Feed_Particle 000012	10.2	1.6	3.4	0.2	0.4	1.6	1.4	2.0	0.0	0.0	0.0	0.5	0.0	0.0	0.0		0.0	1.1	1.3
Feed_Particle 000013	3.2	0.2	2.3	0.1	0.6	0.0	0.6	0.0	0.2	0.0	0.2	0.0	0.0	0.0	0.0		0.0	0.1	0.0
Feed_Particle 000014	0.0	0.2	0.0	0.6	0.1	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.3		0.0	1.6	0.9
Feed_Particle 000015	0.3	0.0	0.1	0.2	0.1	0.0	0.0	0.1	0.0	0.1	0.1	0.4	0.4	0.0	0.0		0.4	0.7	1.2
Feed_Particle 000016	0.1	0.0	0.4	0.0	0.4	0.0	0.6	0.1	1.0	0.3	0.5	0.0	0.3	0.2	0.0		0.0	1.3	2.8
Feed_Particle 000017	10.1	0.8	2.4	0.2	0.5	0.2	0.0	0.0	0.0	0.5	0.1	0.0	0.3	0.5	0.2		0.0	0.1	0.3
Feed_Particle 000018	0.0	0.1	0.1	0.2	0.2	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0		0.0	0.0	0.0
Feed_Particle 000019	0.3	0.0	0.1	0.0	0.2	0.2	0.0	0.0	0.5	0.0	0.0	0.0	0.0	0.2	0.2		0.0	0.0	0.0
Feed_Particle 000020	12.0	0.7	7.6	0.4	2.8	0.3	3.0	2.0	1.6	0.1	0.2	0.2	0.4	0.0	0.3		0.0	1.8	0.0
Feed_Particle 000021	0.0	0.0	0.2	0.4	0.5	0.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2		0.0	1.0	0.0
Feed_Particle 000022	0.0	0.1	0.1	0.0	0.2	0.3	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0		0.0	0.2	1.2
Feed_Particle 000023	22.6	3.8	8.9	0.9	2.3	0.4	2.0	1.1	1.4	0.5	0.4	0.2	0.2	0.2	0.0		0.0	1.6	0.0
Feed_Particle 000024	0.5	0.1	0.5	0.1	0.4	0.0	1.1	1.3	0.4	0.0	0.0	0.0	0.0	0.0	0.0		2.5	3.6	2.0
Feed_Particle 000025	1.8	0.1	0.2	0.2	0.8	0.0	0.2	0.2	0.0	0.6	1.2	0.3	0.4	0.0	0.0		0.0	0.0	1.2
Feed_Particle 000026	1.5	0.2	0.9	0.0	1.1	1.5	0.9	0.0	1.8	0.8	0.6	0.0	0.2	0.1	0.5		2.8	0.6	0.0
Feed_Particle 000027	0.8	0.3	0.1	0.2	0.0	0.0	0.6	1.0	0.5	0.4	0.0	0.0	0.0	0.2	0.0		0.0	0.6	0.0
Feed_Particle 000028	3.8	1.2	2.6	0.0	1.2	1.3	2.5	2.5	2.2	0.7	1.2	0.8	0.4	0.0	0.6		0.0	0.0	0.0
Feed_Particle 000029	19.0	1.8	9.8	0.5	1.6	0.0	1.6	0.7	0.7	0.8	0.0	0.4	0.3	0.0	0.3		0.0	0.0	4.3
Feed_Particle 000030	2.9	0.7	2.5	1.7	2.0	1.8	1.8	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0		1.2	0.0	0.0
Feed_Particle 000031	8.6	1.4	4.0	0.2	1.0	1.1	1.1	0.9	0.2	0.0	0.0	0.0	0.0	0.0	0.2		0.0	0.0	0.0
Feed_Particle 000032	5.0	1.1	2.7	0.6	0.8	0.5	0.9	1.4	0.4	0.2	0.0	0.8	0.4	0.1	0.0		0.0	0.4	0.0
Feed_Particle 000033	0.5	0.0	0.2	0.1	0.5	0.0	0.6	0.0	0.0	0.3	0.3	0.6	0.0	0.4	0.0		0.0	0.6	0.9
Feed_Particle 000034	9.6	1.9	4.6	0.7	1.3	1.7	1.3	2.0	0.0	0.0	0.0	0.3	0.0	0.0	0.0		0.0	0.4	0.8
Feed_Particle 000035	5.5	0.5	3.3	0.4	1.2	0.7	1.4	1.0	0.3	0.0	0.0	0.0	0.0	0.0	0.0		0.0	0.4	0.0
Feed_Particle 000036	0.2	0.0	0.3	0.6	0.7	0.2	0.3	0.0	0.5	0.1	0.2	0.0	0.8	0.5	0.0		0.0	0.9	0.0
Feed_Particle 000037	0.0	0.0	1.2	0.0	0.0	0.4	0.0	0.0	0.0	0.0	0.5	0.6	0.0	0.2	0.8		1.0	2.4	1.9
Feed_Particle 000038	16.5	2.4	9.5	0.6	3.0	1.0	2.1	0.5	0.1	0.0	0.8	0.0	0.3	0.5	0.0		0.0	0.7	0.0
Feed_Particle 000039	0.3	0.3	0.5	0.0	0.5	0.3	2.2	0.7	1.6	0.0	0.2	0.0	0.2	0.4	0.0		0.0	0.0	1.2
Feed_Particle 000040	18.9	3.1	13.7	1.4	4.0	0.0	4.0	1.6	1.4	0.0	0.0	0.0	0.2	0.0	0.0		0.0	0.0	0.0
Feed_Particle 000041	7.0	1.4	5.5	0.1	1.8	1.4	2.7	1.9	0.4	0.0	0.0	0.3	0.0	0.4	0.0		0.0	0.0	1.9
Feed_Particle 000042	0.0	0.0	0.1	0.4	0.4	0.4	0.0	0.0	0.0	0.0	0.0	0.0	0.6	0.0	0.0		0.0	0.0	0.2
Feed_Particle 000043	1.3	0.2	1.2	1.3	1.9	0.0	0.0	2.3	1.6	2.2	0.0	0.2	0.2	0.0	0.3		0.3	0.0	3.1
Feed_Particle 000044	12.2	2.1	8.6	0.0	2.1	1.6	3.6	2.8	0.9	0.0	0.0	0.5	1.1	0.6	2.4		0.0	0.0	0.0
Feed_Particle 000045	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.3	0.4	0.3	1.1	0.2	0.7	0.0	0.1		0.0	3.5	0.0
Feed_Particle 000046	0.8	0.5	0.7	0.1	0.5	0.2	0.5	1.3	0.0	0.3	0.7	1.2	0.0	0.2	0.0		0.0	0.4	0.4
Feed_Particle 000047	1.5	0.7	0.6	0.4	0.9	0.7	2.1	0.9	3.0	0.0	2.3	1.7	1.4	0.1	0.0	0.0	1.7	2.9	3.6
Feed_Particle 000048	0.1	0.1	0.0	0.1	0.1	0.1	0.0	0.0	0.1	0.1	0.0	0.0	0.2	0.1	0.4		0.0	0.0	0.0

Appendix B Elemental concentrations (%) of the first 48 mineral particles of 1129 particles of the flotation concentrate as result of e-probe analysis

Particles #	O	Na	Mg	Al	Si	P	S	K	Ca	Ti	V	Mn	Fe	Co	Zn	Y	Zr	Nb	Cd	Ba	La	Ce	Pr	Nd	Pm	Sm
A	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
Concentrate_Particle 000001	37.9	3.9	0.0	5.7	15.3			1.3	2.0	0.8			17.8			5.4		6.3			0.0	1.0	0.0	0.0	0.0	0.0
Concentrate_Particle 000002	33.2			0.0	10.0			0.5	0.9		0.0		46.2			0.0					0.3	1.6	0.2	1.0	0.4	0.6
Concentrate_Particle 000003	39.5	0.0		0.0	24.8			0.4	9.3				14.4			2.5					1.4	5.3	0.7	2.6	0.1	1.0
Concentrate_Particle 000004	38.2			0.0	25.1			0.4	7.5				6.9			0.0					0.6	6.8	1.1	3.0	0.8	1.7
Concentrate_Particle 000005	33.6		4.4	0.0	15.7			1.2	10.5				8.2			4.3					1.3	7.2	1.3	4.9	1.1	1.2
Concentrate_Particle 000006	20.6			0.0												72.5					0.0	0.2	0.1	0.3	0.2	0.0
Concentrate_Particle 000007	30.1			0.0	17.6			1.3	1.3				2.5			2.8					6.4	20.6	0.7	8.5	0.7	2.0
Concentrate_Particle 000008	27.4	2.9		0.0	12.7				7.9				1.9			0.0					5.9	23.5	0.8	10.9	0.8	2.7
Concentrate_Particle 000009	34.2		5.4	0.0	15.3	1.9		2.0	4.0				5.0			11.4					1.2	10.2	1.3	4.8	0.4	0.8
Concentrate_Particle 000010	37.3			9.0	16.8				8.2			2.0	6.3			2.5					1.1	5.8	1.3	3.4	0.0	0.9
Concentrate_Particle 000011	33.1		2.9	0.0	7.3			0.7	1.3				50.0			0.0					0.0	0.7	0.2	0.3	0.1	0.3
Concentrate_Particle 000012	32.7		0.0	0.0	6.6			0.3	1.9				55.2			1.0					0.0	0.9	0.2	0.1	0.0	0.0
Concentrate_Particle 000013	33.4		3.8	0.0	7.2			1.0	0.7				50.5			0.0					0.0	0.0	0.0	0.5	0.3	0.3
Concentrate_Particle 000014	37.3			5.5	14.0	4.2			9.6			1.7	6.3			4.8					0.9	5.3	1.2	2.7	0.2	0.8
Concentrate_Particle 000015	30.5		2.5	0.0	13.1			1.8	7.6				9.1			5.5					1.4	9.9	1.1	6.3	0.6	1.6
Concentrate_Particle 000016	32.3		0.0	2.8	4.6			0.3	1.2				54.2			0.4					0.0	0.2	0.1	0.0	0.1	0.0
Concentrate_Particle 000017	37.5				25.1					0.5						25.5					0.0	0.0	0.1	0.9	0.0	0.1
Concentrate_Particle 000018	48.9			0.0	23.4	14.4		2.4	2.6				2.7			0.0					0.1	0.7	0.0	0.0	0.0	0.0
Concentrate_Particle 000019	38.5			0.0	19.6	2.8			1.9				0.8			5.2	26.6				0.0	0.6	0.4	0.4	0.1	0.0
Concentrate_Particle 000020	31.6			0.0	17.5			0.4	8.0				2.5			5.7					5.2	16.1	0.8	7.0	0.5	1.9
Concentrate_Particle 000021	35.0			0.0	20.5				8.3			3.7	7.4			0.0					2.0	9.0	0.8	4.3	0.0	1.4
Concentrate_Particle 000022	39.4			0.0	17.3	4.7			5.3				2.9			0.0	28.1				0.1	0.3	0.0	0.0	0.0	0.0
Concentrate_Particle 000023	38.5			10.2	17.4				8.9			1.1	9.1			2.9					0.5	5.9	0.8	3.0	0.0	0.6
Concentrate_Particle 000024	24.0		0.0	0.0	9.7				20.1							11.1					3.8	13.4	0.0	9.9	0.8	3.6
Concentrate_Particle 000025	27.1		1.9	0.0	10.0			0.4	13.9				2.7			4.0					3.9	16.8	2.8	10.2	1.3	2.8
Concentrate_Particle 000026	28.2			0.0	12.7		0.4		7.8				2.9			7.6					6.1	18.1	0.9	5.7	0.3	1.6
Concentrate_Particle 000027	30.9			0.0	0.7		27.7		0.3				19.2			0.4					0.0	0.0	0.1	0.3	0.1	0.1
Concentrate_Particle 000028	34.1		2.8	0.0	18.3				7.7				8.4			0.0					1.4	9.8	0.8	5.7	1.3	2.6
Concentrate_Particle 000029	32.2		3.6	0.0	9.1				0.7				53.6			0.0					0.1	0.1	0.0	0.2	0.1	0.4
Concentrate_Particle 000030	32.4		3.8	0.0	4.4		0.5		0.7				54.1			0.0					0.3	0.4	0.0	0.0	0.1	0.3
Concentrate_Particle 000031	22.6		0.0	0.0	6.1				10.8							12.6					3.4	18.9	0.0	12.2	0.9	3.0
Concentrate_Particle 000032	24.4		1.3	0.0	4.7		2.6		8.4				3.0			3.8					4.7	23.5	1.8	11.6	1.2	3.0
Concentrate_Particle 000033	33.5			0.0	7.4	12.4			0.7				0.8			7.4					6.2	18.0	0.1	7.5	0.5	2.0
Concentrate_Particle 000034	22.9		0.0	0.0	9.6				19.9							8.6					6.3	24.2	0.0	9.6	0.4	1.4
Concentrate_Particle 000035	24.3	3.9	1.0	0.0	6.3				8.3				7.7			3.4					6.4	18.5	1.1	9.2	0.9	3.1
Concentrate_Particle 000036	35.3		2.3	0.0	9.0		5.2	0.4	3.1				26.9			0.0					2.0	8.0	1.2	2.8	0.9	1.3
Concentrate_Particle 000037	33.2		4.8	0.0	16.2			1.0	0.9				3.4			29.0					0.0	0.5	0.3	1.0	0.2	0.5
Concentrate_Particle 000038	30.8		4.3		13.6				1.2				1.4			33.1					0.0	1.9	0.3	0.9	0.2	0.8
Concentrate_Particle 000039	28.5	0.0		0.0	12.8			0.8	11.9				1.5			10.1					2.0	9.3	1.9	7.1	0.8	3.1
Concentrate_Particle 000040	30.1		3.0	0.0	14.1			0.5	11.2				3.5			3.4				7.5	1.9	7.0	3.3	4.3	0.1	1.4
Concentrate_Particle 000041	19.7		0.0	0.0	3.5				7.4				1.6			8.6					5.5	25.0	0.8	12.2	0.8	3.8
Concentrate_Particle 000042	32.7			0.0	7.6				1.4				52.4			0.0					0.0	0.6	0.1	0.0	0.3	0.0
Concentrate_Particle 000043	37.6		1.1	0.0	15.3	4.4		1.0	1.6				1.6			0.0	32.8			0.5	0.0	0.0	0.0	0.6	0.3	0.5
Concentrate_Particle 000044	34.8		0.0	0.0	10.1				8.4	4.0	0.1		38.8			0.0					0.0	0.5	0.2	0.4	0.0	0.0
Concentrate_Particle 000045	35.7		0.9	0.0	20.0				8.3			1.6	10.8			2.5					0.9	5.5	0.4	3.3	0.0	1.2
Concentrate_Particle 000046	40.5			0.0	26.8			0.5	7.1				10.6			0.0					1.0	6.3	0.2	3.0	0.0	0.7
Concentrate_Particle 000047	37.8	0.0		0.0	23.1				7.7			2.0	9.7			3.4					0.6	5.7	0.8	3.6	0.1	1.1
Concentrate_Particle 000048	31.2	5.6		0.0	16.0			0.4	8.7				2.2			4.6					3.3	13.8	0.0	6.7	0.7	2.5

Appendix B Elemental concentrations (%) of the first 48 mineral particles of the flotation concentrate as result of e-probe analysis (continued)

Particles #	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Hf	Ta	Au	Tl	Pb	Th	U
A	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42
Concentrate_Particle 000001	0.7	0.0	0.0	0.4	0.2	0.0	0.5	0.4	0.3	0.0					0.1	0.0
Concentrate_Particle 000002	0.6	0.4	1.0	0.0	0.7	0.0	0.9	0.0	0.5	0.0					0.8	0.0
Concentrate_Particle 000003	1.1	0.0	0.3	0.0	0.0	0.0	0.0	0.0	0.5	0.0	0.3				0.0	0.0
Concentrate_Particle 000004	3.2	0.7	2.6	0.0	0.6	0.1	0.4	0.0	0.2	0.2					0.0	0.0
Concentrate_Particle 000005	0.0	1.2	2.3	0.0	0.0	0.0	1.0	0.0	0.0	0.0					0.0	0.5
Concentrate_Particle 000006	0.2	0.0	0.0	1.7	0.5	2.1	0.5	1.1	0.0	0.0					0.0	0.0
Concentrate_Particle 000007	1.1	2.2	1.1	0.6	0.0	0.0	0.2	0.0	0.1	0.0					0.1	0.0
Concentrate_Particle 000008	0.0	1.3	0.3	0.7	0.0	0.0	0.0	0.0	0.0	0.0					0.2	0.3
Concentrate_Particle 000009	0.0	0.2	1.0	0.3	0.0	0.0	0.0	0.3	0.0	0.0					0.1	0.0
Concentrate_Particle 000010	1.1	1.5	2.2	0.0	0.0	0.0	0.2	0.0	0.0	0.0					0.4	0.0
Concentrate_Particle 000011	0.4	0.3	0.0	0.0	0.4	0.0	0.6	0.2	0.0	0.2					0.1	0.9
Concentrate_Particle 000012	0.0	0.0	0.0	0.0	0.0	0.0	0.4	0.0	0.0	0.1					0.0	0.6
Concentrate_Particle 000013	0.5	0.0	0.8	0.0	0.5	0.0	0.4	0.0	0.0	0.0					0.0	0.0
Concentrate_Particle 000014	1.0	0.9	0.9	0.7	0.3	0.4	0.2	0.1	0.4	0.0					0.0	0.7
Concentrate_Particle 000015	0.5	2.2	3.2	0.0	0.0	0.0	0.7	0.0	0.0	0.0					0.7	2.0
Concentrate_Particle 000016	0.5	0.0	0.4	0.0	0.6	0.0	0.7	0.0	0.0	0.1					0.0	1.7
Concentrate_Particle 000017	0.4	1.3	0.0	3.7	1.1	2.1	0.4	0.8	0.4	0.0		0.3			0.0	0.0
Concentrate_Particle 000018	0.0	0.8	1.5	0.4	0.0	0.2	0.3	0.5	0.0	0.3					0.1	0.7
Concentrate_Particle 000019	0.0	0.0	0.4	0.3	0.0	0.1	0.2	0.0	0.4	0.3					0.9	0.5
Concentrate_Particle 000020	0.9	1.1	0.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0					0.1	0.0
Concentrate_Particle 000021	3.1	1.5	1.3	1.3	0.0	0.0	0.0	0.0	0.2	0.2					0.0	0.0
Concentrate_Particle 000022	0.0	0.2	1.0	0.1	0.0	0.0	0.0	0.4	0.0	0.3					0.0	0.0
Concentrate_Particle 000023	1.0	0.9	0.5	0.7	0.0	0.4	0.5	0.0	0.0	0.0					0.1	0.0
Concentrate_Particle 000024	0.6	3.9	0.0	0.6	0.4	0.6	0.6	0.0	0.2	0.4					0.3	0.0
Concentrate_Particle 000025	0.9	1.8	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.3					0.0	0.0
Concentrate_Particle 000026	0.9	1.8	1.2	0.6	0.0	0.4	0.7	0.0	0.5	0.2					0.0	1.4
Concentrate_Particle 000027	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.1	0.0					0.0	0.0
Concentrate_Particle 000028	3.2	0.6	2.0	0.0	0.8	0.0	0.1	0.0	0.2	0.2					0.0	0.0
Concentrate_Particle 000029	0.4	0.0	0.0	0.0	0.0	0.0	0.0	0.6	0.4	0.5					0.8	1.0
Concentrate_Particle 000030	0.5	0.0	0.0	0.0	0.0	0.0	0.9	0.4	0.5	0.5					0.0	0.1
Concentrate_Particle 000031	1.5	3.3	0.0	1.1	0.8	1.1	0.0	0.0	0.3	0.0					0.6	0.7
Concentrate_Particle 000032	0.5	2.8	2.1	0.5	0.0	0.0	0.0	0.0	0.0	0.0					0.4	0.0
Concentrate_Particle 000033	0.4	1.1	0.4	0.3	0.0	0.4	0.0	0.0	0.2	0.0					0.6	0.0
Concentrate_Particle 000034	1.2	2.8	0.0	0.6	0.0	0.0	0.0	0.0	0.0	0.0					0.5	0.0
Concentrate_Particle 000035	0.0	2.1	1.8	0.0	0.0	0.0	0.1	0.0	0.6	0.3					0.4	0.7
Concentrate_Particle 000036	0.7	0.0	0.5	0.0	0.5	0.2	0.5	0.0	0.0	0.0					0.1	1.2
Concentrate_Particle 000037	0.3	1.6	1.2	2.0	0.9	1.3	0.2	0.5	0.0	0.2					0.0	1.0
Concentrate_Particle 000038	0.6	2.4	0.7	3.4	0.1	1.7	0.6	0.7	0.3	0.0					0.9	0.0
Concentrate_Particle 000039	1.5	3.7	1.7	1.6	0.0	0.3	0.5	0.5	0.2	0.0					0.0	0.0
Concentrate_Particle 000040	0.3	1.8	2.0	0.2	0.0	0.0	0.4	0.4	0.1	0.0					1.2	2.4
Concentrate_Particle 000041	1.7	2.9	1.0	1.8	0.4	0.9	0.5	0.7	0.4	0.0					1.1	0.0
Concentrate_Particle 000042	0.4	0.0	0.5	1.2	0.2	0.0	1.0	0.5	0.1	0.4					0.5	0.1
Concentrate_Particle 000043	0.4	0.0	0.5	0.0	0.4	0.2	0.5	0.2	0.1	0.5					0.0	0.0
Concentrate_Particle 000044	0.4	0.0	0.0	0.0	0.0	0.0	0.7	0.4	0.5	0.1					0.4	0.0
Concentrate_Particle 000045	1.2	1.6	1.3	1.0	0.2	0.0	0.3	0.0	0.5	0.6					0.0	2.4
Concentrate_Particle 000046	1.5	1.0	0.0	0.0	0.0	0.4	0.0	0.0	0.0	0.3					0.0	0.0
Concentrate_Particle 000047	1.2	1.2	0.4	1.4	0.0	0.0	0.0	0.0	0.2	0.0					0.0	0.0
Concentrate_Particle 000048	0.0	2.0	0.7	0.7	0.0	0.0	0.0	0.0	0.0	0.4					0.7	0.0

Appendix C Compound concentrations (%) of the first 48 mineral particles of the feed ore after data batch processing

Particles #	Area, uM^2	Na2O	MgO	Al2O3	SiO2	P2O5	SO3	K2O	CaO	TiO2	V2O5	MnO	Fe2O3	CoO	Cu2O	ZnO		As2O3	Y2O3	ZrO2	Nb2O5	MoO3	CdO	SnO2	BaO	La2O3	Ce2O3	Pr2O3	Nd2O3	Pm2O3
A	B	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15		16	17	18	19	20	21	22	23	24	25	26	27	28
[Feed]_Particle 000001	206.88	2	3	0	32	0	0	0	10	0		3	12		0	0		0	1	1	1	1		0		2	10	1	3	1
[Feed]_Particle 000002	4.02	8	1	0	5	0	0	0	18	0		0	1		0	0		0	5	3	2	3		0		6	20	3	11	1
[Feed]_Particle 000003	1.22	0	0	0	22	0	0	0	28	44	0	0	3		0	0		0	0	1	0	2		0		0	0	0	0	0
[Feed]_Particle 000004	1.85	0	6	0	25	0	0	0	20	28	0	1	14		0	0		0	0	0	1	0		0		0	0	0	0	0
[Feed]_Particle 000005	2.12	3	17	0	19	0	0	0	0	0		0	12		0	0		0	1	1	1	0		0		7	19	3	8	1
[Feed]_Particle 000006	1.59	2	2	0	29	0	1	1	6	0		0	10		0	1		0	0	4	3	7		1		2	6	0	2	1
[Feed]_Particle 000007	8.31	0	0	3	6	0	0	0	3	38	0	21	8		0	0		0	0	0	0	0		0		0	1	1	0	0
[Feed]_Particle 000008	2.49	0	0	0	0	0	0	0	2	42	0	31	13		0	0		0	1	0	0	0		0		1	1	1	0	1
[Feed]_Particle 000009	183.48	2	5	0	29	0	0	0	18	0		2	14		0	0		0	0	3	0	2		0		3	13	2	6	0
[Feed]_Particle 000010	4.08	3	0	0	50	11	0	0	0	0		0	1		0	0		0	0	27	0	1		0		0	0	0	0	0
[Feed]_Particle 000011	18.06	3	3	0	33	0	0	0	13	0		1	16		0	1		0	3	0	1	0		0		2	11	1	3	0
[Feed]_Particle 000012	50.83	2	3	0	30	0	0	0	9	0		3	15		0	0		0	1	0	1	2		1		4	12	2	4	0
[Feed]_Particle 000013	1.59	0	0	0	84	0	0	0	3	0		0	0		0	0		0	1	0	1	0		0		2	4	0	3	0
[Feed]_Particle 000014	1.85	10	2	0	16	0	0	0	1	0		0	68		0	0		0	0	0	3	0		1		0	0	0	0	1
[Feed]_Particle 000015	1.64	1	0	0	66	8	0	0	0	0		0	0		0	0		0	2	15	1	1		0		0	0	0	0	0
[Feed]_Particle 000016	5.4	2	0	0	51	3	0	0	1	0		0	1		0	0		0	6	19	6	0		0		0	0	0	1	0
[Feed]_Particle 000017	2.38	0	0	0	71	0	0	0	1	0		0	0		0	0		0	0	0	0	2		0		3	12	1	3	0
[Feed]_Particle 000018	689.16	0	4	0	0	0	23	0	0	0		0	25		0	0		50	0	0	0	0		0		0	0	0	0	0
[Feed]_Particle 000019	1.75	5	4	0	48	0	0	0	0	0		0	39		0	0		0	1	0	0	0		0		0	0	0	0	0
[Feed]_Particle 000020	7.31	5	3	0	16	0	0	1	8	0		0	2		0	0		0	3	1	1	0		0		3	14	1	9	0
[Feed]_Particle 000021	2.49	0	0	0	0	0	0	0	0	5	0	0	89		0	0		0	0	1	0	0		0		0	0	0	0	0
[Feed]_Particle 000022	19.27	0	0	0	0	0	0	0	0	5	0	0	91		0	0		0	0	0	0	0		0		0	0	0	0	0
[Feed]_Particle 000023	42.25	4	0	0	16	0	0	0	14	0		0	1		0	1		0	4	0	1	0		0		6	26	4	10	1
[Feed]_Particle 000024	1.27	16	1	0	27	0	0	0	3	6		1	9		0	0		0	3	2	11	2		0		0	1	0	1	0
[Feed]_Particle 000025	12.55	3	0	0	30	13	0	0	4	0		0	4		0	0		0	2	29	2	3		0		0	2	0	0	0
[Feed]_Particle 000026	1.91	11	4	0	31	0	0	1	3	0		1	4		0	0		1	9	1	18	1		0		0	2	0	1	0
[Feed]_Particle 000027	2.91	5	1	0	8	0	0	0	6	0		0	70		0	0		0	1	1	1	0		0		1	1	0	0	0
[Feed]_Particle 000028	5.82	2	3	0	31	3	1	0	7	0		2	8		0	0		0	4	12	2	0		0		0	4	1	3	0
[Feed]_Particle 000029	8.15	10	1	0	18	1	0	0	11	0		0	1		0	1		0	2	2	0	1		0		3	22	2	11	1
[Feed]_Particle 000030	5.93	5	5	0	31	0	0	1	3	1		0	6		0	0		0	7	9	12	0		0		2	3	1	3	2
[Feed]_Particle 000031	1.85	2	0	0	70	0	0	0	0	0		0	1		0	0		0	1	0	0	0		0		2	10	2	5	0
[Feed]_Particle 000032	5.88	2	0	0	62	0	0	0	6	0		1	7		0	0		0	0	1	1	1		0		2	6	1	3	1
[Feed]_Particle 000033	22.82	2	1	0	10	0	0	0	1	0		0	69		0	0		0	3	2	6	0		0		0	1	0	0	0
[Feed]_Particle 000034	3.12	4	5	0	37	0	0	0	5	0		2	13		0	0		0	0	0	2	0		0		2	11	2	5	1
[Feed]_Particle 000035	15.3	2	2	0	54	0	0	0	10	0		1	10		0	0		0	1	0	0	0		0		0	6	1	4	0
[Feed]_Particle 000036	6.46	5	0	0	43	8	1	0	0	0		0	2		0	0		0	4	29	0	0		0		1	0	0	0	1
[Feed]_Particle 000037	1.11	3	0	0	51	0	0	0	0	2		5	3		0	0		0	3	3	20	0		0		0	0	0	1	0
[Feed]_Particle 000038	1.8	11	1	0	22	0	0	0	10	0		1	1		0	0		0	2	2	1	0		0		4	19	3	11	1
[Feed]_Particle 000039	6.04	16	4	0	48	0	0	0	0	0		0	4		0	0		0	15	1	0	1		0		0	0	0	1	0
[Feed]_Particle 000040	8.42	6	0	0	15	0	0	0	12	0		1	1		0	0		0	4	2	0	0		0		3	22	4	16	2
[Feed]_Particle 000041	61.95	4	0	0	32	0	0	0	16	0		1	14		0	0		0	0	3	0	0		0		1	8	2	6	0
[Feed]_Particle 000042	3.55	2	0	0	38	14	1	0	1	0		0	1		0	0		0	2	33	2	4		0		0	0	0	0	0
[Feed]_Particle 000043	4.24	5	0	0	56	1	1	0	1	6		1	5		0	0		0	0	2	1	0		1		1	2	0	1	2
[Feed]_Particle 000044	1.48	0	4	0	12	0	0	4	1	0		0	8		0	0		0	0	3	20	2		0		0	14	2	10	0
[Feed]_Particle 000045	3.87	3	2	0	24	17	1	1	1	0		0	3		0	0		0	4	29	4	4		0		0	0	0	0	0
[Feed]_Particle 000046	1.64	0	0	7	18	0	0	0	17	15	0	0	10		0	0		0	0	3	21	0		0		0	1	1	1	0
[Feed]_Particle 000047	2.44	3	0	0	10	0	0	0	2	1		0	2		0	0		0	25	0	26	2		0		1	2	1	1	0
[Feed]_Particle 000048	35.05	0	0	1	0	0	48	0	0	0		0	24		0	0		0	0	1	1	0		0		0	0	0	0	0

Appendix C Compound concentrations (%) of the first 48 mineral particles of the feed ore after data batch processing (continued)

Particles #	Sm2O3	Eu2O3	Gd2O3	Tb2O3	Dy2O3	Ho2O3	Er2O3	Tm2O3	Yb2O3	Lu2O3	HfO2	Ta2O5	Au	PbO	ThO2	UO2
A	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44
(Feed)_Particle 000001	3	2	3	3	0	0	0	0	0	0		0		0	0	0
(Feed)_Particle 000002	2	1	4	2	1	0	0	0	0	1		0		0	0	3
(Feed)_Particle 000003	0	0	0	0	0	0	0	0	0	0		0		1	0	0
(Feed)_Particle 000004	0	1	0	0	0	0	0	0	0	0		0		0	1	0
(Feed)_Particle 000005	2	0	2	2	0	0	0	0	0	0		0		0	0	1
(Feed)_Particle 000006	0	0	1	3	0	0	2	2	2	2		0		0	3	9
(Feed)_Particle 000007	0	3	3	4	0	0	0	0	0	0		0		0	1	0
(Feed)_Particle 000008	1	0	1	2	1	0	0	0	0	0		0		1	1	0
(Feed)_Particle 000009	1	1	1	2	0	0	0	0	0	0		0		0	0	3
(Feed)_Particle 000010	0	0	0	0	0	0	0	0	0	0		0		0	2	2
(Feed)_Particle 000011	2	1	1	2	0	0	0	0	0	0		0		0	0	1
(Feed)_Particle 000012	1	2	2	2	0	0	0	1	0	0		0		0	1	1
(Feed)_Particle 000013	1	0	1	0	0	0	0	0	0	0		0		0	0	0
(Feed)_Particle 000014	0	0	0	0	0	0	0	0	0	0		0		0	2	1
(Feed)_Particle 000015	0	0	0	0	0	0	0	0	0	0		0		0	1	1
(Feed)_Particle 000016	0	0	1	0	1	0	1	0	0	0		0		0	1	3
(Feed)_Particle 000017	1	0	0	0	0	1	0	0	0	1		0		0	0	0
(Feed)_Particle 000018	0	0	0	0	0	0	0	0	0	0		0		0	0	0
(Feed)_Particle 000019	0	0	0	0	1	0	0	0	0	0		0		0	0	0
(Feed)_Particle 000020	3	0	3	2	2	0	0	0	0	0		0		0	2	0
(Feed)_Particle 000021	1	1	0	0	0	0	0	0	0	0		0		0	1	0
(Feed)_Particle 000022	0	0	0	0	0	0	0	0	0	0		0		0	0	1
(Feed)_Particle 000023	3	1	2	1	2	1	0	0	0	0		0		0	2	0
(Feed)_Particle 000024	0	0	1	2	0	0	0	0	0	0		0		3	4	2
(Feed)_Particle 000025	1	0	0	0	0	1	1	0	0	0		0		0	0	1
(Feed)_Particle 000026	1	2	1	0	2	1	1	0	0	0		1		3	1	0
(Feed)_Particle 000027	0	0	1	1	1	0	0	0	0	0		0		0	1	0
(Feed)_Particle 000028	1	2	3	3	3	1	1	1	0	0		1		0	0	0
(Feed)_Particle 000029	2	0	2	1	1	1	0	0	0	0		0		0	0	3
(Feed)_Particle 000030	2	2	2	0	0	0	0	0	0	0		0		1	0	0
(Feed)_Particle 000031	1	1	1	1	0	0	0	0	0	0		0		0	0	0
(Feed)_Particle 000032	1	1	1	2	0	0	0	1	0	0		0		0	0	0
(Feed)_Particle 000033	1	0	1	0	0	0	0	1	0	0		0		0	1	1
(Feed)_Particle 000034	2	2	1	2	0	0	0	0	0	0		0		0	0	1
(Feed)_Particle 000035	1	1	2	1	0	0	0	0	0	0		0		0	0	0
(Feed)_Particle 000036	1	0	0	0	1	0	0	0	1	1		0		0	1	0
(Feed)_Particle 000037	0	0	0	0	0	0	1	1	0	0		1		1	3	2
(Feed)_Particle 000038	3	1	2	1	0	0	1	0	0	1		0		0	1	0
(Feed)_Particle 000039	1	0	2	1	2	0	0	0	0	0		0		0	0	1
(Feed)_Particle 000040	3	0	3	2	2	0	0	0	0	0		0		0	0	0
(Feed)_Particle 000041	2	2	3	2	0	0	0	0	0	0		0		0	0	2
(Feed)_Particle 000042	0	1	0	0	0	0	0	0	1	0		0		0	0	0
(Feed)_Particle 000043	2	0	0	3	2	3	0	0	0	0		0		0	0	4
(Feed)_Particle 000044	2	2	4	3	1	0	0	1	1	1		3		0	0	0
(Feed)_Particle 000045	0	0	0	0	0	0	1	0	1	0		0		0	4	0
(Feed)_Particle 000046	1	0	1	2	0	0	1	1	0	0		0		0	0	0
(Feed)_Particle 000047	1	1	2	1	3	0	3	2	2	0		0	0	2	3	4
(Feed)_Particle 000048	0	0	0	0	0	0	0	0	0	0		0		0	0	0

Appendix D Compound concentrations (%) of the first 48 mineral particles of the flotation concentrate after data batch processing

Particles #	Area, (MP2)	Na2O	MgO	Al2O3	SiO2	P2O5	SO3	K2O	CaO	TiO2	V2O5	MnO	Fe2O3	CoO	CuO	ZnO	As2O3	SeO2	Y2O3	ZrO2	Nb2O5	MoO3	CdO	BaO	La2O3	Ce2O3	Pr2O3
A	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
Concentrate_Particle 000001	1.9	5.3	0.0	10.7	32.8			1.6	2.8	1.4			25.5						6.8		9.0				0.0	1.2	0.0
Concentrate_Particle 000002	3.1			0.0	21.4			0.6	1.3		0.0		66.1						0.0						0.4	1.9	0.2
Concentrate_Particle 000003	3.6	0.0		0.0	53.0			0.5	7.4				20.6						3.2						1.6	6.2	0.9
Concentrate_Particle 000004	12.6			0.0	53.6			0.5	10.4				9.9						0.0						0.7	7.9	1.3
Concentrate_Particle 000005	1.5		7.4	0.0	33.6			1.4	14.7				11.7						5.5						1.5	8.5	1.6
Concentrate_Particle 000006	37.2			0.0															92.1						0.0	0.3	0.1
Concentrate_Particle 000007	2.4			0.0	37.6			1.6	1.8				3.6						3.6						7.5	24.1	0.9
Concentrate_Particle 000008	13.6	3.9		0.0	27.1				11.1				2.7						0.0						6.9	27.5	0.9
Concentrate_Particle 000009	1.5		9.0	0.0	32.8	4.3		2.4	5.6				7.2						14.5						1.4	12.0	1.5
Concentrate_Particle 000010	7.3			16.9	36.0				11.5			2.6	9.0						3.1						1.2	6.8	1.5
Concentrate_Particle 000011	1.5		4.8	0.0	15.5			0.8	1.9				71.5						0.0						0.0	0.9	0.2
Concentrate_Particle 000012	1.3		0.0	0.0	14.2			0.3	2.6				78.9						1.3						0.0	1.0	0.3
Concentrate_Particle 000013	9.1		6.3	0.0	15.4			1.2	1.0				72.2						0.0						0.0	0.0	0.0
Concentrate_Particle 000014	3.8			10.3	30.0	9.6			13.4			2.2	9.0						6.1						1.1	6.2	1.4
Concentrate_Particle 000015	4.6		4.2	0.0	28.1			2.2	10.6				13.0						6.7						1.7	11.6	1.3
Concentrate_Particle 000016	6.6		0.0	5.3	9.8			0.4	1.7				77.4						0.5						0.0	0.2	0.2
Concentrate_Particle 000017	18.7				58.7					0.8									32.4						0.0	0.0	0.2
Concentrate_Particle 000018	2.1			0.0	50.1	33.1		2.9	3.7				3.9						0.0						0.1	0.8	0.0
Concentrate_Particle 000019	5.6			0.0	42.0	6.4			2.6				1.1						6.6	35.9					0.0	0.7	0.5
Concentrate_Particle 000020	4.9			0.0	37.4			0.5	11.2				3.5						7.3						6.1	18.9	0.9
Concentrate_Particle 000021	54.4			0.0	43.8				11.6			4.7	10.5						0.0						2.3	10.5	0.9
Concentrate_Particle 000022	2.1			0.0	37.0	10.8			7.4				4.1						0.0	38.0					0.1	0.3	0.0
Concentrate_Particle 000023	1.6			19.2	37.2				8.3			1.4	13.0						3.6						0.6	7.0	0.9
Concentrate_Particle 000024	14.6		0.0	0.0	12.3				28.1										14.0						4.4	15.7	0.0
Concentrate_Particle 000025	1.5		3.1	0.0	21.5			0.5	18.1				3.9						5.1						4.6	19.6	3.2
Concentrate_Particle 000026	9.5			0.0	27.3		1.1		10.9				4.1						9.6						7.2	21.2	1.1
Concentrate_Particle 000027	20.4			0.0	1.4		69.2		0.4				27.4						0.5						0.0	0.0	0.1
Concentrate_Particle 000028	39.1		4.6	0.0	39.1				10.8				12.0						0.0						1.7	11.5	0.9
Concentrate_Particle 000029	1.5		6.0	0.0	11.0				1.0				76.6						0.0						0.1	0.2	0.0
Concentrate_Particle 000030	3.4		6.3	0.0	9.4		1.3		1.0				77.4						0.0						0.3	0.5	0.0
Concentrate_Particle 000031	16.4		0.0	0.0	13.1				15.1										16.0						4.0	22.2	0.0
Concentrate_Particle 000032	8.4		2.1	0.0	10.0		6.5		11.7				4.3						4.8						5.5	27.6	2.1
Concentrate_Particle 000033	17.2			0.0	15.9	28.4			1.0				1.2						9.4						7.3	21.1	0.1
Concentrate_Particle 000034	4.2		0.0	0.0	11.9				22.2										10.9						7.4	28.3	0.0
Concentrate_Particle 000035	4.0	5.2	1.6	0.0	13.6				11.6				10.9						4.3						7.6	21.6	1.2
Concentrate_Particle 000036	1.3		3.8	0.0	19.2		12.9	0.4	4.3				38.5						0.0						2.3	7.1	1.4
Concentrate_Particle 000037	5.5		8.0	0.0	34.6			1.2	1.2				4.8						36.9						0.0	0.6	0.3
Concentrate_Particle 000038	7.8		7.1		29.2				1.7				2.0						42.1						0.0	2.2	0.4
Concentrate_Particle 000039	16.3	0.0		0.0	27.4			1.0	16.7				2.2						12.9						2.3	10.9	2.2
Concentrate_Particle 000040	1.8		4.9	0.0	30.1			0.6	15.6				5.1						4.3					8.4	2.2	8.2	3.9
Concentrate_Particle 000041	37.5		0.0	0.0	7.4				10.3				2.2						11.0						6.4	29.2	1.0
Concentrate_Particle 000042	5.4			0.0	16.3				2.0				74.9						0.0						0.0	0.6	0.1
Concentrate_Particle 000043	2.7		1.8	0.0	32.7	10.1		1.2	2.2				2.4						0.0	44.3				0.5	0.0	0.0	0.0
Concentrate_Particle 000044	2.0		0.0	0.0	21.5				11.7	6.7	0.2		55.5						0.0						0.0	0.7	0.2
Concentrate_Particle 000045	13.8		1.5	0.0	42.7				11.6			2.1	15.5						3.1						1.0	6.5	0.5
Concentrate_Particle 000046	2.2			0.0	57.3			0.6	10.1				15.1						0.0						1.2	7.4	0.3
Concentrate_Particle 000047	8.0	0.0		0.0	49.5				10.8			2.6	13.8						4.3						0.7	6.6	0.9
Concentrate_Particle 000048	9.3	7.5		0.0	34.3			0.5	12.2				3.2						5.8						3.9	16.1	0.0

Appendix D Compound concentrations (%) of the first 48 mineral particles of the flotation concentrate after data batch processing (continued)

Particles #	Nd2O3	Pm2O3	Sm2O3	Eu2O3	Gd2O3	Tb2O3	Dy2O3	Ho2O3	Er2O3	Tm2O3	Yb2O3	Lu2O3	HfO2	Ta2O5	Au	PbO	BiO	ThO2	UO2
A	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	46
Concentrate_Particle 000001	0.0	0.0	0.0	0.8	0.0	0.0	0.4	0.3	0.0	0.5	0.4	0.4	0.0						0.2	0.0
Concentrate_Particle 000002	1.2	0.5	0.6	0.7	0.5	1.2	0.0	0.9	0.0	1.0	0.0	0.6	0.0						0.9	0.0
Concentrate_Particle 000003	3.0	0.2	1.1	1.2	0.0	0.4	0.0	0.0	0.0	0.0	0.0	0.5	0.0	0.4					0.0	0.0
Concentrate_Particle 000004	3.5	0.9	2.0	3.7	0.9	3.0	0.0	0.7	0.2	0.5	0.0	0.2	0.2						0.0	0.0
Concentrate_Particle 000005	5.7	1.3	1.4	0.0	1.4	2.7	0.0	0.0	0.0	1.1	0.0	0.0	0.0						0.0	0.6
Concentrate_Particle 000006	0.4	0.2	0.0	0.2	0.0	0.0	2.0	0.5	2.4	0.5	1.2	0.0	0.0						0.0	0.0
Concentrate_Particle 000007	9.9	0.9	2.3	1.2	2.6	1.3	0.7	0.0	0.0	0.2	0.0	0.1	0.0						0.1	0.0
Concentrate_Particle 000008	12.7	0.9	3.1	0.0	1.5	0.3	0.8	0.0	0.0	0.0	0.0	0.0	0.0						0.2	0.3
Concentrate_Particle 000009	5.6	0.5	1.0	0.0	0.3	1.2	0.4	0.0	0.0	0.0	0.4	0.0	0.0						0.2	0.0
Concentrate_Particle 000010	4.0	0.0	1.1	1.3	1.7	2.5	0.0	0.0	0.0	0.3	0.0	0.0	0.0						0.4	0.0
Concentrate_Particle 000011	0.3	0.1	0.4	0.4	0.3	0.0	0.0	0.5	0.0	0.7	0.2	0.0	0.3						0.1	1.0
Concentrate_Particle 000012	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.5	0.0	0.0	0.1						0.0	0.7
Concentrate_Particle 000013	0.6	0.3	0.4	0.6	0.0	0.9	0.0	0.6	0.0	0.4	0.0	0.0	0.0						0.0	0.0
Concentrate_Particle 000014	3.2	0.2	1.0	1.2	1.0	1.0	0.8	0.3	0.5	0.2	0.1	0.5	0.0						0.0	0.8
Concentrate_Particle 000015	7.3	0.7	1.9	0.6	2.6	3.7	0.0	0.0	0.0	0.8	0.0	0.0	0.0						0.8	2.2
Concentrate_Particle 000016	0.0	0.2	0.0	0.6	0.0	0.4	0.0	0.7	0.0	0.7	0.0	0.0	0.1						0.0	1.9
Concentrate_Particle 000017	1.0	0.0	0.1	0.5	1.6	0.0	4.2	1.2	2.4	0.4	0.9	0.4	0.0		0.3				0.0	0.0
Concentrate_Particle 000018	0.0	0.0	0.0	0.0	0.9	1.7	0.4	0.0	0.2	0.3	0.6	0.0	0.4						0.1	0.8
Concentrate_Particle 000019	0.5	0.1	0.0	0.0	0.0	0.5	0.4	0.0	0.1	0.2	0.0	0.4	0.4						1.0	0.6
Concentrate_Particle 000020	8.2	0.6	2.2	1.0	1.2	0.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0						0.1	0.0
Concentrate_Particle 000021	5.1	0.0	1.7	3.6	1.8	1.5	1.4	0.0	0.0	0.0	0.0	0.3	0.2						0.0	0.0
Concentrate_Particle 000022	0.0	0.0	0.0	0.0	0.3	1.2	0.1	0.0	0.0	0.0	0.4	0.0	0.3						0.0	0.0
Concentrate_Particle 000023	3.5	0.0	0.7	1.1	1.0	0.6	0.8	0.0	0.4	0.6	0.0	0.0	0.0						0.1	0.0
Concentrate_Particle 000024	11.6	0.9	4.2	0.7	4.5	0.0	0.7	0.4	0.7	0.7	0.0	0.3	0.5						0.3	0.0
Concentrate_Particle 000025	11.8	1.5	3.2	1.1	2.1	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.4						0.0	0.0
Concentrate_Particle 000026	6.7	0.4	1.8	1.1	2.0	1.4	0.7	0.0	0.4	0.8	0.0	0.6	0.2						0.0	1.6
Concentrate_Particle 000027	0.3	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.2	0.1	0.1						0.0	0.0
Concentrate_Particle 000028	6.6	1.5	3.1	3.7	0.7	2.3	0.0	0.9	0.0	0.1	0.0	0.2	0.3						0.0	0.0
Concentrate_Particle 000029	0.2	0.2	0.4	0.5	0.0	0.0	0.0	0.0	0.0	0.0	0.7	0.5	0.6						1.0	1.1
Concentrate_Particle 000030	0.0	0.1	0.3	0.6	0.0	0.0	0.0	0.0	0.0	1.0	0.4	0.6	0.6						0.0	0.1
Concentrate_Particle 000031	14.2	1.1	3.5	1.7	3.8	0.0	1.3	0.9	1.2	0.0	0.0	0.3	0.0						0.7	0.8
Concentrate_Particle 000032	13.5	1.4	3.4	0.5	3.2	2.4	0.5	0.0	0.0	0.0	0.0	0.0	0.0						0.4	0.0
Concentrate_Particle 000033	8.8	0.6	2.3	0.5	1.2	0.5	0.3	0.0	0.4	0.0	0.0	0.2	0.0						0.7	0.0
Concentrate_Particle 000034	11.2	0.5	1.6	1.4	3.2	0.0	0.7	0.0	0.0	0.0	0.0	0.0	0.0						0.6	0.0
Concentrate_Particle 000035	10.8	1.1	3.6	0.0	2.4	2.1	0.0	0.0	0.0	0.2	0.0	0.7	0.3						0.4	0.7
Concentrate_Particle 000036	3.3	1.1	1.5	0.8	0.0	0.6	0.0	0.5	0.2	0.6	0.0	0.0	0.0						0.2	1.4
Concentrate_Particle 000037	1.1	0.2	0.6	0.4	1.8	1.4	2.3	1.0	1.4	0.3	0.6	0.0	0.2						0.0	1.1
Concentrate_Particle 000038	1.0	0.2	0.9	0.7	2.8	0.9	3.9	0.2	1.9	0.7	0.8	0.3	0.0						1.0	0.0
Concentrate_Particle 000039	8.3	0.9	3.5	1.7	4.2	2.0	1.9	0.0	0.3	0.6	0.6	0.2	0.0						0.0	0.0
Concentrate_Particle 000040	5.0	0.1	1.6	0.4	2.0	2.3	0.2	0.0	0.0	0.5	0.4	0.1	0.0						1.3	2.7
Concentrate_Particle 000041	14.2	0.9	4.4	1.9	3.3	1.1	2.0	0.4	1.0	0.6	0.8	0.5	0.0						1.3	0.0
Concentrate_Particle 000042	0.0	0.3	0.0	0.5	0.0	0.6	1.4	0.3	0.0	1.1	0.5	0.1	0.5						0.6	0.1
Concentrate_Particle 000043	0.7	0.3	0.6	0.4	0.0	0.6	0.0	0.5	0.3	0.6	0.2	0.1	0.6						0.0	0.0
Concentrate_Particle 000044	0.5	0.0	0.0	0.5	0.0	0.0	0.0	0.0	0.0	0.8	0.5	0.6	0.2						0.4	0.0
Concentrate_Particle 000045	3.9	0.0	1.4	1.4	1.8	1.5	1.1	0.3	0.0	0.3	0.0	0.5	0.7						0.0	2.7
Concentrate_Particle 000046	3.5	0.1	0.8	1.7	1.2	0.0	0.0	0.0	0.4	0.0	0.0	0.0	0.3						0.0	0.0
Concentrate_Particle 000047	4.2	0.1	1.3	1.4	1.4	0.5	1.6	0.0	0.0	0.0	0.0	0.2	0.0						0.0	0.0
Concentrate_Particle 000048	7.8	0.8	2.9	0.0	2.3	0.8	0.8	0.0	0.0	0.0	0.0	0.0	0.5						0.8	0.0

Appendix E REE contents in the feed ore, concentrate, and tailings of gravity separation test No.1

Products	Mass, g	Weight (%)	Y Grade (ppm)	Y (Units)	La Grade (ppm)	La (Units)	Ce Grade (ppm)	Ce Units	Pr Grade (ppm)	Pr Units
A	*B*	*C*	*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*
C1	23.6	3.2	6240	147264.0	6730	158828.0	16750	395300.0	2180	51448.0
C2	18.5	2.5	14500	268250.0	12100	223850.0	30800	569800.0	4160	76960.0
C3	18.9	2.6	13400	253260.0	10800	204120.0	28100	531090.0	3830	72387.0
C4	14.9	2.0	8730	130077.0	6510	96999.0	17150	255535.0	2360	35164.0
Tailings	660.1	89.7	2385	1574338.5	1645	1085864.5	4405	2907740.5	617	407281.7
C1 - C4	75.9	10.3	10525	798851.0	9009	683797.0	23079	1751725.0	3109	235959.0
Feed Calc.	736.0	100.0	3224	2372864.0	2404	1769344.0	6331	4659616.0	874	643264.0

Products	Nd Grade (ppm)	Nd Units	Sm Grade (ppm)	Sm Units	Eu Grade (ppm)	Eu Units	Gd Grade (ppm)	Gd Units	Tb Grade (ppm)	Tb Units	Dy Grade (ppm)
A	*9*	*10*	*11*	*12*	*13*	*14*	*15*	*16*	*17*	*18*	*19*
C1	8680	204848.0	1870	44132.0	174.5	4118.2	1620	38232.0	262.0	6183.2	1355
C2	16850	311725.0	3860	71410.0	349.0	6456.5	3260	60310.0	544.0	10064.0	3000
C3	15600	294840.0	3620	68418.0	325.0	6142.5	3040	57456.0	507.0	9582.3	2760
C4	9780	145722.0	2310	34419.0	210.0	3129.0	2000	29800.0	333.0	4961.7	1825
Tailings	2580	1703058.0	618	407941.8	58.0	38285.8	544	359094.4	90.0	59409.0	489
C1 - C4	12610	957135.0	2877	218379.0	261	19846.2	2448	185798.0	406	30791.2	2198
Feed Calc.	3614	2659904.0	851	626336.0	79.0	58144.0	740	544640.0	123.0	90528.0	665

Appendix E REE contents in the feed ore, concentrate, and tailings of gravity separation test No.1 (continued)

Products	Dy Units	Ho Grade (ppm)	Ho Units	Er Grade (ppm)	Er Units	Tm Grade (ppm)	Tm Units	Yb Grade (ppm)	Yb Units	Lu Grade (ppm)	Lu Units
A	*20*	*21*	*22*	*23*	*24*	*25*	*26*	*27*	*28*	*29*	*30*
C1	31978.0	238.0	5616.8	538	12696.8	62.0	1463.2	302	7127.2	28.7	677.3
C2	55500.0	549.0	10156.5	1310	24235.0	155.5	2876.8	777	14374.5	73.9	1367.2
C3	52164.0	505.0	9544.5	1195	22585.5	141.5	2674.4	723	13664.7	68.0	1285.2
C4	27192.5	333.0	4961.7	799	11905.1	94.0	1400.6	479	7137.1	45.4	676.5
Tailings	322788.9	88.0	58088.8	211	139281.1	25	16502.5	130	85813.0	12.5	8251.3
C1 - C4	166834.5	399	30279.5	941	71422.4	111	8414.9	557	42303.5	53	4006.1
Feed Calc.	489440.0	120.0	88320.0	286	210496.0	34	25024.0	174	128064.0	17	12512.0

Appendix F REE contents in the feed ore, concentrate, and tailings of gravity separation test No.2

Products	Mass, g	Weight (%)	Y Grade (ppm)	Y (Units)	La Grade (ppm)	La (units)	Ce Grade (ppm)	Ce Units	Pr Grade (ppm)	Pr Units	Nd Grade (ppm)	Nd Units
A	*B*	*C*	*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*	*10*
Conc.	22.3	3.4	17750	395825.0	17100	381330.0	43900	978970.0	5800	129340.0	22900	510670.0
Middl.	31.6	4.9	4830	152628.0	3860	121976.0	10300	325480.0	1390	43924.0	5760	182016.0
Tail	594.1	91.7	2605	1547630.5	1875	1113937.5	5000	2970500.0	694	412305.4	2895	1719919.5
Conc.+ Mid	53.9	8.3	10175	548453.0	9338	503306.0	24201	1304450.0	3215	173264.0	12851	692686.0
Feed Calc.	648.0		3235	2096280.0	2496	1617408.0	6597	4274856.0	904	585792.0	3723	2412504.0
Feed	649		3560		2550		6800		946		3890	

Products	Sm Grade (ppm)	Sm Units	Eu Grade (ppm)	Eu Units	Gd Grade (ppm)	Gd Units	Tb Grade (ppm)	Tb Units	Dy Grade (ppm)	Dy Units
A	*11*	*12*	*13*	*14*	*15*	*16*	*17*	*18*	*19*	*20*
Conc.	4900	1092700	456	10168.8	4250	94775.0	704	15699.2	3790	84517.0
Middl.	1350	42660.0	120.5	3807.8	1135	35866.0	185.5	5861.8	993	31378.8
Tail	688	4087408	64.2	38141.2	592	3517072	99.8	59291.2	540	320814.0
Conc.+ Mid	2819	151930.0	259	13976.6	2424	130641.0	400	21561.0	2150	115895.8
Feed Calc.	865	560520.0	80	51840.0	744	482112.0	125.0	81000.0	674	436752.0
Feed	922		83.6		779		130		716	

Appendix F REE contents in the feed ore, concentrate, and tailings of gravity separation test No.2 (continued)

Products	Ho Grade (ppm)	Ho Units	Er Grade (ppm)	Er Units	Tm Grade (ppm)	Tm Units	Yb Grade (ppm)	Yb Units	Lu Grade (ppm)	Lu Units
A	*21*	*22*	*23*	*24*	*25*	*26*	*27*	*28*	*29*	*30*
Conc.	678	15119.4	1605	35791.5	185.0	4125.5	925	20627.5	86.6	1931.2
Middl.	177.5	5609.0	416	13145.6	48.4	1529.4	247	7805.2	23.7	748.9
Tail	96.7	57449.5	229	136048.9	27.2	16159.5	138.5	82282.9	13.6	8079.8
Conc.+ Mid	385	20728.4	908	48937.1	105	5654.9	528	28432.7	50	2680.1
Feed Calc.	121.0	78408.0	285	184680.0	34	22032.0	171	110808.0	17	11016.0
Feed	129		308		36.8		186.5		17.8	

Appendix G REE contents in the feed ore, concentrate, and tailings of gravity separation test No.3

Products	Weight (g)	Weight (%)	Y Grade (ppm)	Y (units)	La Grade (ppm)	La (units)	Ce Grade (ppm)	Ce (units)	Pr Grade (ppm)	Pr (units)	Nd Grade (ppm)	Nd (units)
A	*B*	*C*	*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*	*10*
C1	69.8	9.1%	10200	711960	8970	626106	23400	1633320	3110	217078	12450	869010
C2	65.5	8.5%	6070	397585	4640	303920	12450	815475	1690	110695	6830	447365
C3	61	7.9%	3500	213500	2580	157380	7010	427610	1010	61610	4060	247660
Tailings	574.6	74.5%	2490	1430754	1695	973947	4560	2620176	672	386131.2	2670	1534182
Feed Calc.	770.9	100.0%	3572	2753799	2674	2061353	7130	5496581	1006	775514.2	4019	3098217
Feed	769		3560		2550		6800		946		3890	
C1-C3	196.3	25.5%	6740	1323045	5540	1087406	14653	2876405	1984	389383	7968	1564035

Products	Sm Grade (ppm)	Sm (units)	Eu Grade (ppm)	Eu (units)	Gd Grade (ppm)	Gd (units)	Tb Grade (ppm)	Tb (units)	Dy Grade (ppm)	Dy (units)
A	*11*	*12*	*13*	*14*	*15*	*16*	*17*	*18*	*19*	*20*
C1	2930	204514	273	19055.4	2560	178688	393	27431.4	2100	146580
C2	1645	107747.5	152	9956	1430	93665	222	14541	1235	80892.5
C3	969	59109	94.5	5764.5	891	54351	138	8418	748	45628
Tailings	656	376937.6	63.4	36429.64	599	344185.4	94	54012.4	517	297068.2
Feed Calc.	971	748308.1	92	71205.54	870	670889.4	135	104402.8	740	570168.7
Feed	922		84		779		130		716	
C1-C3	1892	371371	177	34776	1664	326704	257	50390	1391	273101

Appendix G REE contents in the feed ore, concentrate, and tailings of gravity separation test No.3 (continued)

Products	Ho Grade (ppm)	Ho (units)	Er Grade (ppm)	Er (units)	Tm Grade (ppm)	Tm (units)	Yb Grade (ppm)	Yb (units)	Lu Grade (ppm)	Lu (units)
A	21	22	23	24	25	26	27	28	29	30
C1	376	26244.8	874	61005.2	102.5	7154.5	495	34551	53	3699.4
C2	222	14541	525	34387.5	62.2	4074.1	309	20239.5	33.2	2174.6
C3	134	8174	321	19581	38.5	2348.5	189.5	11559.5	20.6	1256.6
Tailings	94.3	54184.78	221	126986.6	26.4	15169.44	131	75272.6	14.4	8274.24
Feed Calc.	134	103144.58	314	241960.3	37	28746.54	184	141622.6	20	15404.84
Feed	129		308		37		186		18	
C1-C3	249	48960	586	114974	69	13577	338	66350	36	7131

Appendix H REE contents in the feed ore, concentrate, and tailings of gravity separation test No.4

Products	Weight (g)	Weight (%)	Y Grade (ppm)	Y (units)	La Grade (ppm)	La (units)	Ce Grade (ppm)	Ce (units)	Pr Grade (ppm)	Pr (units)	Nd Grade (ppm)	Nd (units)
A	*B*	*C*	*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*	*10*
C1	27.9	3.9%	19150	534285	17650	492435	46300	1291770	5400	150660	24000	669600
C2	22.2	3.1%	8370	185814	6950	154290	18500	410700	2480	55056	9860	218892
C3	20.7	2.9%	4850	100395	3860	79902	10550	218385	1445	29911.5	6090	126063
Middlings	108.4	15.3%	3380	366392	2580	279672	7010	759884	991	107424.4	3890	421676
Tailings	527.8	74.7%	2480	1308944	1685	889343	4560	2406768	666	3515148	2680	1414504
Feed Calc.	707.0	100.0%	3530	2495830	2681	1895642	7196	5087507	982	694566.7	4032	2850735
Feed	706.0		3560		2550		6800		946		3890	
C1-C3	70.8	10.0%	11589	820494	10263	726627	27131	1920855	3328	235628	14330	1014555

Products	Sm Grade (ppm)	Sm (units)	Eu Grade (ppm)	Eu (units)	Gd Grade (ppm)	Gd (units)	Tb Grade (ppm)	Tb (units)	Dy Grade (ppm)	Dy (units)
A	*11*	*12*	*13*	*14*	*15*	*16*	*17*	*18*	*19*	*20*
C1	4900	136710	523	14591.7	4700	131130	761	21231.9	4020	112158
C2	2380	52836	222	4928.4	2080	46176	319	7081.8	1730	38406
C3	1455	30118.5	139	2877.3	1325	27427.5	203	4202.1	1090	22563
Middlings	919	99619.6	85.5	9268.2	810	87804	122	13224.8	669	72519.6
Tailings	659	347820.2	63	33251.4	602	317735.6	93.2	49190.96	517	272872.6
Feed Calc.	944	667104.3	92	64917	863	610273.1	134	94931.56	733	518519.2
Feed	922		84		779		130		716	
C1-C3	3103	219665	316	22397	2892	204734	459	32516	2445	173127

Appendix H REE contents in the feed ore, concentrate, and tailings of gravity separation test No.4 (continued)

Products	Ho Grade (ppm)	Ho (units)	Er Grade (ppm)	Er (units)	Tm Grade (ppm)	Tm (units)	Yb Grade (ppm)	Yb (units)	Lu Grade (ppm)	Lu (units)
A	*21*	*22*	*23*	*24*	*25*	*26*	*27*	*28*	*29*	*30*
C1	745	20785.5	1755	48964.5	204	5691.6	995	27760.5	104.5	2915.55
C2	306	6793.2	718	15939.6	85	1887	419	9301.8	44.6	990.12
C3	194	4015.8	450	9315	53	1097.1	262	5423.4	28.3	585.81
Middlings	119.5	12953.8	279	30243.6	32.9	3566.36	164	17777.6	17.8	1929.52
Tailings	93.2	49190.96	220	116116	26.3	13881.14	130.5	68877.9	14	7389.2
Feed Calc.	133	93739.26	312	220578.7	37	26123.2	183	129141.2	20	13810.2
Feed	129		308		37		186		18	
C1-C3	446	31595	1048	74219	123	8676	600	42486	63	4491

Appendix I REE contents in the feed ore, concentrate, and tailings of WHIMS test

Products	Weight (g)	Weight (%)	Y Grade (ppm)	Y (units)	La Grade (ppm)	La (units)	Ce Grade (ppm)	Ce (units)	Pr Grade (ppm)	Pr (units)	Nd Grade (ppm)	Nd (units)
A	*B*	*C*	*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*	*10*
C1	5.6	11.6%	2290	12824	1290	7224	3450	19320	501	2805.6	1880	10528
C2	3.1	6.4%	4240	13144	2280	7068	6260	19406	909	2817.9	3490	10819
C3	3.4	7.1%	5120	17408	3020	10268	8250	28050	1200	4080	4650	15810
C4	2.8	5.8%	6280	17584	4070	11396	11050	30940	1615	4522	6170	17276
C5	2.9	6.0%	5490	15921	3930	11397	10700	31030	1530	4437	5930	17197
Tailings	30.3	63.0%	1550	46965	1100	33330	3090	93627	426	12907.8	1635	49540.5
Calc. Feed	48.1	100.0%	2575	123846	1677	80683	4623	222373	656	31570.3	2519	121170.5
C1 - C5	17.8	37%	4319	76881	2660	47353	7233	128746	1048	18662.5	4024	71630

Products	Sm Grade (ppm)	Sm (units)	Eu Grade (ppm)	Eu (units)	Gd Grade (ppm)	Gd (units)	Tb Grade (ppm)	Tb (units)	Dy Grade (ppm)	Dy (units)
A	*11*	*12*	*13*	*14*	*15*	*16*	*17*	*18*	*19*	*20*
C1	479	2682.4	51.5	288.4	492	2755.2	84	470.4	471	2637.6
C2	863	2675.3	89.6	277.76	895	2774.5	156.5	485.15	863	2675.3
C3	1145	3893	118	401.2	1160	3944	197	669.8	1075	3655
C4	1510	4228	146.5	410.2	1465	4102	247	691.6	1305	3654
C5	1410	4089	137.5	398.75	1335	3871.5	217	629.3	1145	3320.5
Tailings	391	11847.3	35.8	1084.74	353	10695.9	56.8	1721.04	310	9393
Calc. Feed	612	29415	59	2861.05	585	28143.1	97	4667.29	527	25335.4
C1 - C5	987	17567.7	100	1776.31	980	17447.2	166	2946.25	896	15942.4

Appendix I REE contents in the feed ore, concentrate, and tailings of WHIMS test (continued)

Products	Ho Grade (ppm)	Ho (units)	Er Grade (ppm)	Er (units)	Tm Grade (ppm)	Tm (units)	Yb Grade (ppm)	Yb (units)	Lu Grade (ppm)	Lu (units)
A	*21*	*22*	*23*	*24*	*25*	*26*	*27*	*28*	*29*	*30*
C1	84	470.4	210	1176	23.3	130.48	117	655.2	11.85	66.36
C2	150	465	366	1134.6	41.2	127.72	196	607.6	18.2	56.42
C3	186.5	634.1	444	1509.6	48.5	164.9	231	785.4	23.4	79.56
C4	223	624.4	528	1478.4	58.3	163.24	277	775.6	26.2	73.36
C5	196	568.4	461	1336.9	52.7	152.83	260	754	26.2	75.98
Tailings	54.5	1651.35	141.5	4287.45	16.65	504.495	86.8	2630.04	9.02	273.306
Calc. Feed	92	4413.65	227	10922.95	26	1243.665	129	6207.84	13	624.986
C1 - C5	155	2762.3	373	6635.5	42	739.17	201	3577.8	20	351.68

Appendix J REE contents in the feed ore, concentrate, and tailings of flotation test

Products	Weight (g)	Weight (%)	La Grade (ppm)	La (units)	Ce Grade (ppm)	Ce (units)	Pr Grade (ppm)	Pr (units)	Nd Grade (ppm)	Nd (units)	Sm Grade (ppm)	Sm (units)
A	*B*	*C*	*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*	*10*
C1	63.6	8.7%	4010	255036	10850	690060	1445	91902	5890	374604	1355	86178
C2	23.9	3.3%	2390	57121	6560	156784	874	20888.6	3580	85562	851	20338.9
C3	33.7	4.6%	8950	301615	22800	768360	2950	99415	12300	414510	2720	91664
C4	51.2	7.0%	6970	356864	18350	939520	2440	124928	9750	499200	2240	114688
Tailings	559.7	76.5%	1710	957087	4870	2725739	639	357648.3	2620	1466414	694	388431.8
Feed Calc.	732.1	100.0%	2633	1927723	7213	5280463	949	694781.9	3880	2840290	958	701300.7
Feed			2680		7210		1020		3890		925	
C1-C4	172.4	23.5%	5630	970636	14819	2554724	1956	337133.6	7969	1373876	1815	312868.9

Products	Eu Grade (ppm)	Eu (units)	Gd Grade (ppm)	Gd (units)	Tb Grade (ppm)	Tb (units)	Dy Grade (ppm)	Dy (units)	Ho Grade (ppm)	Ho (units)
A	*11*	*12*	*13*	*14*	*15*	*16*	*17*	*18*	*19*	*20*
C1	125	7950	1150	73140	174.5	11098.2	879	55904.4	152	9667.2
C2	81.1	1938.29	757	18092.3	121.5	2903.85	671	16036.9	124.5	2975.55
C3	241	8121.7	2280	76836	328	11053.6	1540	51898	266	8964.2
C4	198.5	10163.2	1840	94208	275	14080	1370	70144	231	11827.2
Tailings	62.7	35093.19	611	341976.7	106	59328.2	600	335820	106.5	59608.05
Feed Calc.	86	63266.38	825	604253	134	98463.85	724	529803.3	127	93042.2
Feed	91.8		856		146.5		771		133.5	
C1-C4	163	28173.19	1521	262276.3	227	39135.65	1125	193983.3	194	33434.15

Appendix J REE contents in the feed ore, concentrate, and tailings of flotation test (continued)

Products	Er Grade (ppm)	Er (units)	Tm Grade (ppm)	Tm (units)	Yb Grade (ppm)	Yb (units)	Lu Grade (ppm)	Lu (units)	Y Grade (ppm)	Y (units)
A	*21*	*22*	*23*	*24*	*25*	*26*	*27*	*28*	*29*	*30*
C1	332	21115.2	36.3	2308.68	194.5	12370.2	18.5	1176.6	4220	268392
C2	283	6763.7	31.9	762.41	171.5	4098.85	17.05	407.495	3290	78631
C3	569	19175.3	59.8	2015.26	294	9907.8	27.1	913.27	7630	257131
C4	502	25702.4	53.3	2728.96	262	13414.4	24.3	1244.16	6600	337920
Tailings	265	148320.5	33.5	18749.95	166.5	93190.05	17.25	9654.825	2830	1583951
Feed Calc.	302	221077.1	36	26565.26	182	132981.3	18	13396.35	3450	2526025
Feed	314		38.3		188		18.6		3590	
C1-C4	422	72756.6	45	7815.31	231	39791.25	22	3741.525	5464	942074

Appendix K REE contents in the feed ore and residue of direct leaching test

Products	Weight (g)	Weight (%)	Y Grade (ppm)	La Grade (ppm)	Ce Grade (ppm)	Pr Grade (ppm)	Nd Grade (ppm)	Sm Grade (ppm)	Eu Grade (ppm)
A	*B*	*C*	*1*	*2*	*3*	*4*	*5*	*6*	*7*
Residue	3.8	76.0%	1245	271	757	103.5	418	117	13.8
Feed	5.0	100.0%	3420	2570	7060	919	3770	891	82.9

Products	Gd Grade (ppm)	Tb Grade (ppm)	Dy Grade (ppm)	Ho Grade (ppm)	Er Grade (ppm)	Tm Grade (ppm)	Yb Grade (ppm)	Lu Grade (ppm)
A	*8*	*9*	*10*	*11*	*12*	*13*	*14*	*15*
Residue	150.5	32.3	204	44.1	119	16.05	91.7	10.25
Feed	809	131.5	719	133	309	37.2	186	17.9

Appendix L REE contents in the feed ore and residue of Leach after Flotation test

Products	Weight (g)	Weight (%)	Y Grade (ppm)	La Grade (ppm)	Ce Grade (ppm)	Pr Grade (ppm)	Nd Grade (ppm)	Sm Grade (ppm)	Eu Grade (ppm)
A	*B*	*C*	*1*	*2*	*3*	*4*	*5*	*6*	*7*
Residue	3.3	66.0%	1680	220	585	80.8	316	109	13.3
Feed	5.0	100.0%	4220	4010	10850	1445	5890	1355	125

Products	Gd Grade (ppm)	Tb Grade (ppm)	Dy Grade (ppm)	Ho Grade (ppm)	Er Grade (ppm)	Tm Grade (ppm)	Yb Grade (ppm)	Lu Grade (ppm)
A	*8*	*9*	*10*	*11*	*12*	*13*	*14*	*15*
Residue	166	39.7	261	53.3	139.5	18.9	104.5	10.9
Feed	1150	174.5	879	152	332	36.3	194.5	18.5